10.0 Beneficiation

10.1 INTRODUCTION

Iron is the world's most widely used metal as well as the fourth most abundant (5% of the earth crust) rock-forming element. Iron ore deposits containing up to 70% Fe have been formed by transformation of a small portion of the crustal iron by igneous, sedimentary and metamorphic processes. Iron ore deposits are widely distributed throughout the world. The largest concentration of iron ore deposits is found in banded sedimentary iron ore formations of pre-Cambrian age, constituting bulk of the world's largest iron ore reserves and consisting mainly of fine-grained iron oxides (hematite, martite, goethite, limonite) associated with quartz, silicates and iron carbonates. Massive deposits of magnetite, sometimes with hematite, supposed to be of igneous origin, are also important iron ore deposits in many parts of the world.

With the exception of meteorites, iron never occurs in native state[1]. Iron ores occur mostly as oxides (magnetite, hematite, goethite and limonite) and to a lesser degree as carbonates. Iron ores like other minerals seldom occur in the purity and size composition conforming to user's specifications. In many cases, presence of deleterious impurities like silica, alumina, sulphur and phosphorus beyond desired limits render the ores unsuitable for use in various industries.

Iron ores are mostly used in production of iron and steel and direct reduced iron (DRI). Iron ores are also used in production of ferro-alloys, cement, etc. Very high grade iron ore concentrate is used for ferrite production for subsequent use in electronic and electrical industries. Small quantity of high grade magnetite concentrate is used as dense medium in coal washing plants. Owing to the demand for iron ores of specific chemical composition and physical structure, almost all iron ores mined today need certain treatment (preparation, beneficiation, agglomeration, etc.) to make them suitable for various applications.

10.1.1 Ore Treatment Methods

High grade ores (Fe > 65% and all other impurities within limits) are usually subjected to multistage crushing and screening to obtain coarse (-40 +10 mm) and fine (-10 mm) products. Coarse product is directly fed to blast furnace for iron making. The fine product is further processed, employing mechanical classifiers, hydrocyclones, etc. to obtain -10 mm +100 mesh and -100 mesh products which constitute the feed to sintering and pelletisation, respectively. Low grade ores with higher percentage of impurities require complex and expensive treatment. The treatment methods depend on physical, chemical and mineralogical characteristics of ore. Oxide ores containing coarse crystalline hematite grains do not require any fine grinding for liberation of ore minerals from associated gangue minerals and are usually beneficiated by gravity methods. Earthy ores containing fine-grained hematite, limonite, goethite, etc. need froth flotation, wet high intensity magnetic separation, electrostatic separation, selective flocculation, etc. to produce marketable products. Low intensity magnetic separation methods are used to beneficiate oxide ores containing magnetite. Ores containing substantial amounts of limonite, goethite and siderite (iron carbonate) are usually calcined before use.

10.1.1.1 Ore Preparation

Iron ores extracted from mines are generally in the form of lumps up to 1200 mm. The high grade ore lumps need to be reduced to suitable size before use. The various ore preparation methods adopted for obtaining suitable feed for iron making are as follows :

Crushing : Most of the blast furnaces used for iron making demand iron ore lumps in the size range 10-40 mm with minimum proportion of fines. Therefore, r.o.m. (900-1200 mm) must be reduced to the requisite size by crushing. Since single-stage crushing cannot produce blast furnace feed of required size from the r.o.m., multistage crushing is usually necessary. Depending on the nature of the ore and capacity of the plant, jaw or gyratory crushers are employed for first stage of crushing (primary crushing). Jaw crushers perform better on clayey and plastic material due to greater throw. Gyratory crushers are more suitable for hard and abrasive ores as they tend to give more cubical product from laminated and slabby ore(2). Most open-pit mines discharge the r.o.m. into gyratory crusher from several sides rather than dumping from only one side as is done into pan feeders of jaw crushers. The gyratory crusher capacity is more than double the capacity of a jaw crusher. Hence it is preferred over jaw crusher, provided the crushing capacity required is more than 600 tph. In designing crushing circuit for hard taconite ore, it is advisable to overdesign the primary crushing plant to provide stockpiles of coarse product and thus to ensure steady feed to secondary crushers. The reason for this is that even the best quality crusher liners need replacement every 10-15 days. The largest jaw and gyratory crushers employed in iron ore crushing plants have the feed opening of 1524 mm X 2134 mm and 1524 mm, respectively. The size of the product obtained from primary crusher is in the range of 150-200 mm and power required in the range of 3-4 kWh/t for crushing to 100 mm.

The second stage of crushing (secondary crushing) is usually carried out, employing cone crushers and roll crushers for hard and soft ores, respectively, if the aim is to produce blast furnace feed (25 - 40 mm). Older installations were employing reduction gyratory crushers in a few plants. The cone crushers with mantle diameter up to 2000-2500 mm are in operation in many iron ore processing plants. If ore is very hard, and fine product (10-25 mm) is desired, third stage of crushing (tertiary crushing) is required and usually short-head cone crushers are employed. The power required to obtain a product in the size range of 10-25 mm from 200-150 mm feed is about 4-5 kWh/t. Most of the processing plants generally employ two stages of crushing. If ore is slippery and tough, tertiary crushing is substituted by coarse grinding in rod mills. Tertiary crushing is generally done in closed circuit with screens to ensure that the material is all within the desired size range. The reduction ratio for primary, secondary and tertiary crushers should not exceed 8, 6-8 and 4-6, respectively.

Screening : Screening plays a vital role in successful operation of iron ore processing plants. Very coarse material, usually fed to primary crusher is screened on grizzly. The lumps in the size range of 300-20 mm can be sized on grizzly. Trommels are generally used to handle/size iron ore particles in the range of 50-5 mm, either wet or dry. They are preferred where disintegration of material and washing are required. Vibrating screens are the most important screening device used in iron ore processing. They handle material in the size range of 250 mm to 48 mesh. The most widely used screens for coarse sizing (+40 mm) are mechanically vibrated. Vibrating screens with rectangular mesh are preferred for acicular, moist and clayey ore. Screens with square opening are used for sizing slabby material. Rapi-fine sieve bends have found an increasing application in closed circuit grinding of iron ore to obtain a product containing no oversize material. They can work efficiently up to separation size of 100 microns. Very fine material can be separated by hydrocyclones.

Grinding : In many iron ore deposits, iron ore minerals are closely associated with gangue minerals requiring fine grinding for effective

liberation. Generally, the feed to grinding mills is in the size range of 10-25 mm. Fine grinding is usually done in rod mills in open circuit followed by either ball mills or pebble mills in closed circuit with classifiers, cyclones, screens, etc.

Autogenous Grinding : In recent years, the secondary gyratory crushers, cone crushers and rod mills have been substituted by only one machine which utilises coarse and hard pieces of the ore itself as the grinding media in place of balls, rods, pebbles, etc. Rod and ball wear, a costly affair in conventional grinding, is eliminated. Thus the circuit becomes very simple and involves less labour cost. All these factors make autogenous grinding an ideal choice provided the r.o.m. contains some hard material and rest of the ore is made of crystalline particles held together by a weak cementing material. Labrador specularite ores bear testimony to this. If ore contains no hard material, some steel balls of 100 mm (5- 10% of mill volume) are added. In some instances, as in taconite mills of Western Mesabi and Iron Ore Concentrator at Kudremukh (India), even though the ore contains some hard material, a small ball charge up to 10% of mill volume is used. This type of grinding is known as Semi-Autogenous Grinding (SAG). This enables smaller mills to match the grinding capacity of bigger autogenous mills.

Blending : Blending is usually practiced to even out the variation in physical and chemical quality of iron ore from a single source or used to homogenise the blend of ores received from different sources. This is particularly important for those plants which do not have their captive mines and are partly or fully dependent upon supply of iron ores from various sources. In recent years, blending has earned the distinction of becoming the satisfactory tool for iron ore homogenisation. The ore is laid down in consecutive layers on a number of beds having trapezoidal or triangular cross-section by employing travelling/reversing-boom stacker or gantry. The ore is reclaimed from the beds in a direction at right angle to that in which ore has been laid down. A minimum of two beds are employed. one is laid down whilst the other is being reclaimed. The capacity of blending section may be about 1000 tph with a reclaiming capacity of 500 tph.

Transportation : Transportation costs for iron ore frequently exceed the mining and processing cost as iron ore is a relatively cheap commodity[1]. It is, therefore, essential to select the most economic method of ore transportation after weighing pros and cons of various systems. Water transportation is invariably cheaper than road and rail transportation, provided the ocean transport from deep water ports is available and large ships (2,00,000 tonnes ore carrying capacity)can be used. M/s Marcona Corporation is already shipping iron ore concentrate from Peru to Japan. The concentrate slurry is pumped aboard ship where it is dewatered to a non-fluid consistency. When the cargo reaches its destination, it is repulped and pumped ashore where it is dewatered, pelletized or otherwise processed. This method of transportation eliminates a costly dewatering step at shipping point, provides a dust-free cargo, ensures lower loading and unloading cost, and dispenses with the need for transport of pellets over long distance which results in their partial disintegration.

A notable developement in ore transportation is slurry transport of fine ore concentrate in pipelines. The first such pipeline (225 mm dia) was put into operation in Tasmania (Australia) by the Savage River Iron Ore Company for transporting 2.25 million tonnes of fine magnetite concentrate over a distance of 85 km. It is generally preferred when other modes of transport are cost prohibitive. M/s Kudremukh Iron Ore Company is transporting by slurry pipeline about 7.5 million tonnes of fine iron ore concentrates from Kudremukh to Mangalore over a distance of about 65 km[2]. At Mangalore, slurry is dewatered through filtration. The pulp density of iron ore concentrate slurry for trouble-free transport should be in the range of 60-65% solids and material should be all finer than 100 mesh size.

10.1.2 Beneficiation

The nature of iron minerals and associated gangue minerals decides the method of beneficiation to be adopted. The common methods take advantage of the difference in the following physical properties of iron minerals and associated gangue minerals for bringing about desired improvement in the quality of the ore.

i) Hardness

ii) Specific gravity

iii) Magnetic properties

iv) Electrostatic properties

v) Response to flotation (both anionic and cationic)

10.1.2.1 Washing and Wet Scrubbing

When iron ore consists essentially of coarse and fine granular particles of iron minerals intermixed with either barren sand or loosely adhered clay, method of separation constitutes simple wet scrubbing, screening to obtain coarse and clean iron ore concentrate. Screen undersize are normally fed to mechanical classifiers. The overflow from washing classifiers is the waste product in most of the cases whereas the rake or sand product constitutes fine iron ore concentrate. The processing of Mesabi Range ores, brown residual limonitic ores in some Southern States of the USA, and iron ores from many deposits of Bihar, Goa, Madhya Pradesh, Orissa, etc. in India come under this category of treatment.

10.1.2.2 Gravity Separation

The common iron ore minerals have usually high specific gravity (hematite : 5.1, magnetite : 5.2, goethite : 4.2 and siderite : 3.85) as compared to the most common associated gangue minerals like quartz and chert (2.65), and calcite or limestone (2.7 to 2.75). If iron minerals are structurally free from associated gangue minerals (waste rock), they can be separated by various gravity methods employing different machines. Proper feed preparation is one of the most important steps to ensure effectiveness of gravity separation. The feed preparation may include (i) crushing and grinding to liberation size and also to ensure feed of proper size to a particular machine, (ii) removal of slimes which otherwise increase viscosity of the pulp and affect efficiency of the machines, and (iii) proper sizing of crude fractions before subsequent treatment.

a) Heavy-media separation : For coarse ore (-50 mm +3 mm), heavy- media separators using ferro-silicon suspension in a rotary drum (spiral and drum-type vessels) are most commonly used. For fine ore (-3 mm +28 mesh or up to 65 mesh), heavy-media cyclones are used. The cyclone-type separator utilizes centrifugal as well as gravitational force to make separation between ore and gangue minerals. The centrifugal force makes it possible to bring about separation at a specific gravity lower than that required in the conventional separator and therefore, magnetite in combination with ferro-silicon (up to 0-30%) in place of Ferro-silicon alone can be used as a media in cyclones. One cyclone unit can treat 100-125 tph material. Ferro-silicon (Si : 15 - 16%, Fe : 81-82%, others : 2-5%), mostly finer than 65 mesh size containing 50% -200 mesh material is predominantly used in iron ore heavy-media separation and it gives a slurry having specific gravity up to 3.2 which is sufficient to separate most of the common gangue minerals in float. Magnetite alone gives a slurry of 2.5 specific gravity only and that is why magnetite in combination with ferro-silicon is used as a media in heavy-media cyclones. Due to the lower viscosity of heat treated (atomised) ferro-silicon medium, it is possible to operate at higher specific gravity (3.4 to 3.5). Ferro-silicon loss is the largest single element of cost in heavy-media separation. Normal losses are in the range of 0.1 to 0.4 kg/t of ore treated. Ferro-silicon, being magnetic in nature is easily recovered by magnetic separation. Ferro-silicon losses are accounted mainly (70-80%) by attachment to sink and float particles and to a lesser extent (20-30%) in magnetic separation. When material

to be treated is irregular and porous in nature, ferro-silicon losses due to attachment increases to the extent of 1 to 1.5 kg/t of ore treated. Ore in which iron mineral has a low apparent specific gravity is concentrated using a low specific gravity suspension (3-3.1 : for -50 mm +12 mm feed and 2.7 to 2.8 for a -12 mm +3 mm feed). When the iron mineral is hard hematite associated with non-porous high specific gravity gangue mineral, the specific gravity of suspension may be raised up to 3.45 for coarse fraction and 3.2 for fine fraction. When the ore feed contains both hematite and magnetite, it is necessary to separate magnetite prior to heavy-media separation. This type of ore is being treated at Kaiser Steel Company, Eagle Mountain California (USA). In the USA, two Mesabi Range GM plants and Albama iron ore HM plants are already in operation utilizing heat treated ferro-silicon. Mesabi Range iron ore processing plant is also employing HM cyclones for treating fine-size material.

b) **Spiralling** : Humphrey's spirals have wide range of applications in gravity treatment of iron ores. Since there are no moving parts in Humphrey's spirals and spirals of other make, the concentration is effected by water flow only. Feed to spirals should be in the range of -20 +150 mesh. Recovery of iron is low in -150 +325 mesh and it is practically nil in case of 325 mesh feed. Iron ore processing plants at Quebec, Cartier Mining Company (Canada) at Mesabi Range (USA) and at Kudremukh (India) are handling -10 mesh, -65 mesh and -20 mesh material. Spirals are normally operated at a pulp density of 25-30% solids and capacity of a spiral is in the range of 2-3.5 tph for a double-start spiral depending upon fineness of the feed and nature of cleaning. Water requirement for treating a tonne of sample in roughing and cleaning stages is 5-10 gpm and 8-12 respectively.

c) **Jigging** : Jigging of iron ores is normally done at the feed size (-6 mm +65 mesh) suitable for treatment by HM cyclones. Jig plants have advantage over cyclone plants in respet of capital cost and often substitute cyclones in case reserves are limited or capital is not available for expensive HM cyclones. Iron ores are being processed by jigging at Barsua plant (India), Eagle Mountain California (USA), Minnesota plant (USA), etc. The pulp density of the feed to the jigs ranges between 25 and 35% solids, and jig loading is in the range of 0.6 to 0.8 tph per square feet of jig bed. Hutch water requirements are 10-15 gpm per square feet of jig bed. Recently, a modified version of jig known as Batac jig has been developed in Germany. The largest available Batac jig in the market is of 5m x 6.2 m size with a throughput capacity of 500 tph. This jig is reported to have capability to treat both coarse as well as fine feed. Tests conducted on Indian and Brazilian iron ore employing Batac jig have yielded promising results. Some Brazilian iron ore processing plants are already operating Batac jigs.

d) **Reichert Cone Concentrator** : It is a wet gravity concentrating device designed for high capacity applications (60 to 100 tph). It accepts a high pulp density feed (55 - 70% solids) and can treat material from 3 mm to 30 microns and in no case finer than 20 microns. Therefore, feed to Reichert cone concentrator in Carol Lake Iron Ore Processing Plant in Canada is all +20 micron in size[4]. The -20 mesh material is removed by dry cycloning. Replacement of spirals by cone concentrator in the circuit has improved iron recovery. Magnetite concentrate is also being recovered in New Zealand by employing Reichert cone concentrator.

e) **Multi Gravity Separator (MGS)** : The treatment of fines and ultra-fines poses problem because conventional gravity separators as discussed earlier are inefficient and ineffective to treat them. A new generation of gravity separator known as multi-gravity separator has been developed to recover very fine mineral particles up to 5 micron. Plant trials have been conducted successfully on iron ore from Suydvaranger A/s Norway. About 600 units of Chinese rotary drum concentrator nearly resembling MGS are already in operation in China for the recovery of fine iron and tin

values. This type of separator may prove a boon to Indian iron ore processing industry for the recovery of iron values from slimes and tailings in coming years.

10.1.2.3 Magnetic Separation

Magnetic separators exploit the difference in magnetic properties of ore and gangue minerals and are used to separate valuable magnetic iron minerals from the non-magnetic associated gangue minerals. Magnetic separators are classified into low and high intensity machines which may be further subdivided into dry wet separators.

a) Low intensity separators : When the economic iron ore mineral is magnetite, low intensity separators are almost invariably used. This is because the method is very cheap and effective. Rotary drum separators with permanent magnet or electromagnets are commonly used. The magnetic field in the concentration zone (50 mm from the shell of separator) is of the order of 1,000 - 2,000 gauss. Modern permanent magnets with adequately high field strength, because of their freedom from maintenance and electrical problems are preferred over electromagnets. For treatment of coarse ores (100 to 6mm), dry cobbers of belt or rotary-type are generally used. Fine material (less than 3 or 6mm) is almost invariably concentrated by wet separators. The field strength of magnets treating fine material (-6 mm) is 700-800 gauss for roughing, 600-700 gauss for cleaning and 350-400 gauss for finishing. Double or triple drum separators (0.9 m dia x 1.2 to 2.4 m width) are being employed in most of the plants in the world. Eriec Mining Company, Hoyt Lakes, Minnesota, employs both electromagnets and permanent magnets whereas Butler Taconite Plant, Minnesota employs both dry and wet separators. Griffith Mines and Canada treats +40 mm -40 +30 mm material on magnetic pulleys. Fine material (-325 mesh) is treated on triple drum separators. Savage River Mines, Australia also employs magnetic pulleys for coarse material and drum separators for fine material. Kudremukh iron ore concentrator employs triple and double drum separators in roughing and cleaning stages, respectively.

b) High intensity separators : High intensity magnetic separators are designed with a magnetic intensity of 7,000 - 20,000 gauss in the separation zone to recover feebly magnetic minerals like hematite. Dry separators are employed in Europe but not in North America for concentrating iron ore because ore to be treated must be bone dry and sized.

Wet high intensity magnetic separators are generally used for treating hematitic and limonitic ores. At CVRD, 28 Jones separators have been installed to treat 25 million tpy ore to produce 10 million tpy concentrate.

This type of separator is also being used in Canada to produce blast furnace grade concentrate and super-grade concentrate suitable for direct reduction and melting in electric furnace after pelletisation. It is also being utilized for making ferrite grade concentrate ($SiO_2 < 0.3\%$). Ferox Iron Ltd., after purchasing iron ore concentrate (Magnetite : 2%, SiO_2 : 6%, rest : hematite) from Quebec Cartier Mining Company is upgrading it to ferrite grade by Jones separator after grinding it to -35 mesh. NMDC (India) is also producing ferrite grade super-concentrate by wet high intensity magnetic separation. Feed to this type of separators should be -20 or -48 mesh and the pulp density around 50% solids. This machine required large quantity of high pressure (2.81 kg/sq cm) water. Power consumption is low (0.75 to 1.0 kWh/ton of feed.)

10.1.2.4 Roasting Followed by Magnetic Separation

Non-magnetic iron ores (hematite, limonite and siderite) are rendered magnetic by roasting. Hematite and limonite when subjected to reduction roasting to a temperature of 500-550°C becomes strongly magnetic whereas siderite when roasted to a temperature of 700-775°C in a neutral atmosphere becomes magnetic. To avoid reoxidation, such roasted ores are quenched in water. After roasting, the ore is ground to liberation size and subjected to low intensity magnetic separation. Roasting makes the ore very weak and therefore requires less energy for grinding. However, roasting cost being prohibitive due to increase in cost of petroleum

products, this technique is seldom employed to process iron ores on commercial scale now -a-days.

10.1.2.5 Electrostatic Separation

This type of separation is generally employed to produce high grade concentrate. It works best on dry crystalline non-magnetic iron oxide minerals finer than 10 mesh and coarser than 325 mesh. The mineral surface should be entirely free from slimes and moisture. At Wabush Larador Plant, the spiral concentrate is water-washed to remove all -325 mesh material and is dried at 110°C and then fed to a battery of high tension separators. The concentrate is upgraded from 64 to 66.5% Fe and silica is reduced from 6 to 2.25%. Electrostatic separators can also be used for producing super-grade concentrate suitable for ferrite manufacture.

10.1.2.6 Froth Flotation

Froth flotation is now used in a number of major iron ore processing plants in the world. For effective flotation, the feed should be finer than 65 mesh. Anionic flotation employing fatty acid or petroleum sulphonate collectors is adopted to float out most of the iron oxide minerals leaving behind the gangue minerals (quartz and chert) in the tailings. Though magnetite can also be floated, it is never recovered by flotation as it can be cheaply recovered by low intensity magnetic separation. Crystalline hematite, such as specularite can be effectively floated. Earthy hematite and limonite do not respond well to flotation and hence is not recovered by this method. Anionic flotation is also resorted to for selective flotation of apatite from iron ore by depressing iron minerals with starch. Cationic flotation using amine as collector is adopted for selective flotation of quartz from magnetitic iron ore. Cationic flotation is generally done on a deslimed feed. The ore from Republic Mine, Michigan (USA) assaying 36.5% Fe contains specular hematite as the main iron minerals with minor amounts of magnetite and martite associated with chert. The ground ore is conditioned with anionic collector (fatty acids) at high pulp density (70% solids). The temperature during conditioning is maintained around 45-60°C. The cleaner concentrate assays 65.4% Fe with a wt.% yield of 46%. The ores from Empire Mine of M/s Cleveland Cliffs Iron Co. and Groveland Iron mine of M/s Hanna Mining Co. in USA assay 22.5% Fe and 34.5% Fe, respectively. Magnetite is the main iron mineral in both the ores. The ore from the Empire Mine is upgraded by magnetic separation and cationic flotation of quartz from the magnetic concentrate. The final concentrate assayed 66.5% Fe, 6.48% SiO_2 with wt.% yield of 32.2 The ore from Groveland Mine is upgraded by anionic flotation of iron minerals (hematite) under acidic medium using sodium petroleum sulphonate and fatty acid as collector. Magnetite is recovered by magnetic separation. The final concentrate assayed 64.4% Fe and 6.3% SiO_2 with 42.2 wt.% yield.

10.1.2.7 Selective Flocculation -Desliming-Cum-Froth Flotation

Iron ore from Tilden mine of Cleveland Cliffs Iron Company, Michigan (USA), is very fine-grained hard ore. The main iron minerals, martite, goethite and hematite are intimately associated with chert and quartz, the predominant gangue minerals. The minor gangue minerals are silicates and carbonates. The average size of iron ore mineral grains is in the range of 10-25 micron. The ore is ground to 82% -25 microns size to effect fair liberation. The treatment of such a fine material by conventional flotation technique is a hard nut to crack due to less selectivity in flotation, high reagent consumption and production of uncontrollable barren froth. The process adopted for treating this type of ore comprises fine grinding to liberation size, selective flocculation of iron minerals, desliming of fine cherty materials and cationic flotation of coarse cherty minerals from the deslimed material. Starch is used at a pH of 10.5 to 11 for selective flocculation of iron minerals and sodium silicate as dispersant for siliceous gangue. Sodium hydroxide is used as pH modifier. During cationic flotation of coarse gangue minerals by amine, starch is used as

depressant for iron minerals. By adopting this process, a high grade concentrate assaying 65.6% Fe from a low grade ore (Fe : 35.9%), with 70.2% Fe recovery (wt.% yield : 38.4) can be obtained. The capacity of the plant is 30,000 tpd (11 million tpy) of r.o.m. and 45 million tpy of concentrate. The reagent requirements are 0.57 kg/t sodium silicate, 1.0 kg/t sodium hydroxide, 1.66 (0.47 + 0.56 + 0.65) kg/t starch and 0.25 (0.17 + 0.08) kg/t amine. It is noteworthy that selective flocculation is effective on +5 micron goethite particles and +2 micron hematite particles. If the gangue minerals are calcite, apatite and clay/other silicates, STPP (sodium tripolyphosphate) with controlled dosages needs to be added as dispersant in place of sodium silicate.[8] Excess dosage of STPP may disperse even fine iron ore mineral particles along with gangue minerals (calcite, apatite, etc.). Coarse gangue minerals may be removed from the sand by their anionic flotation at about 12 pH.

10.1.2.8 Column Flotation

Column flotation had gained wide acceptance in iron ore processing industries because of its advantage over conventional cells as they are capable of producing high grade concentrate with low capital and operating costs, and greater selectivity against silica rejects[6] in iron ore industry. Columns are effectively used in treating fine-grained concentrate during final cleaning stages. Samarco Mineracao SA, Brazil has already employed four 3.6 m dia x 13.2 m height and one 2.4 m dia x 10.8 m height flotation columns of Cominco-make for the removal of silica from hematitic ores.

10.1.3 Agglomeration

Treatment of high grade iron ore lumps (multistage crushing, screening, scrubbing, etc.) for obtaining close-sized feed for blast furnace generates substantial amount (25 to 40%) of fines (-10mm).[10] Beneficiation of low grade iron ores to obtain blast furnace/DRI grade concentrates, in most of the cases, produces fine-size concentrate. The fine beneficiated concentrate and -10 mm material generated during feed preparation cannot be used as such in iron making by blast furnace or DRI furnaces. Their presence in blast furnace affects the operation and productivity of the furnaces. The fines clog the space between the lumps and thereby prevent the free flow of reducing gases, besides a large proportion of fines is usually carried off as flue dust. Therefore, It is imperative to agglomerate the high grade ore fines and fine beneficiated concentrates to render them suitable for iron and steel making. Sintering, pelletisation and briquetting are the different processes being adopted for agglomerating iron ore fines.

10.1.3.1 Briquetting

Briquetting is the earliest process for agglomerating iron ore fines. It is defined as the compaction of fine particles by application of high pressure in a more or less close mould to produce agglomerates of suitable size. It can be done hot or cold and with/without binder depending upon the nature of material. Binders are generally necessary to produce brequettes of sufficient strength with application of less pressing force. Fines of any size can be briquetted but pressing force increases with the fineness of the feed. The pressing force is less for soft ores/hot material, generally in the range of 20-40 KN/cm roll-width and more for hard ores (120-160 KN/cm roll-width). Briquettes are produced in the shape of pillow, egg peanut and cylinder. The sizes of the briquettes may vary from 5 to 200 cc depending upon the end use. The different stages of briquetting are feed preparation, pressing/compaction, curing and evaluation of quality of briquettes. Hard ores are crushed to -3 mm size. Mixing of ore fines with water and binders is generally done in trommels. The filling degree of trommels for proper mixing is 60% of its volume. The binders used are molasses, sulphite lye, lime, sodium silicate, starch, etc. Cements and sulphite lye are generally used for briquetting limonite ores, and pitch and tar are used for dust, mill scales, etc. After mixing, the mix is compacted by roll presses. After compaction, briquettes are cured for different length of time. Curing of briquettes allows the chemical action of binder with ore

particles to take place and develops sufficient strength. It also helps in driving of moisture. Generally, briquettes are stacked in piles (2m or more depth) on conveyor belts. After curing, the mechanical properties of briquettes (cold crushing strength, trommel strength, i.e. abrasion resistance, drop strength and bulk density) are determined. Though briquetting plants have low capital cost and briquettes have better metallurgical properties (superior bed permeability, less resistance to gas flow, greater density leading to increase in capacity of blast furnace), the iron ore briquetting has not been able to make much headway because of relatively high processing cost and limited capacity of briquetting machines. However, the briquetting is employed to agglomerate small quantity of mill dust, scales and other circulating material in many German iron & steel making plants. The process has of late acquired great importance in briquetting of DRI produced by Midrex and other gas reduction processes to prevent reoxidation and self combustion. The cold briquetting is generally applied to agglomerate -3 mm sponge iron particles while hot briquetting process is applied for agglomerating the entire quantity of material emerging from the gas-based direct reduction furnaces without using any binder. Hot briquetting is particularly applicable where it is necessary to store or transport sponge iron otherwise they are susceptible to reoxidation and self combustion. Most of the gas-based direct reduction plants all over the world including India employ briquetting for agglomerating spong iron fines.

10.1.3.2 Sintering

Sintering has been the traditional method for agglomerating iron ore fines (-10 mm +100 mesh). In this, ore fines after mixing properly with mill scale, blue dust, return fines, coke breeze (-3mm) flux (-3mm : lime, dolomite, olivine, serpentine, etc) and moisture are charged onto a fixed/continuous horizontal travelling grate and the top of bed is ignited at high temperature (1200 to 1300°C) by oil, or gas burners for few minutes (10-20% of total sintering time). After that, burners are shut off, air is continuously drawn downwards throughout the length of grate by suction fan so that flame front gradually travels down through the bed of the sinter mix. This combustion raises the temperature inside the bed to 1250-1600°C depending upon the amount of fuel and suction developed. The iron minerals are reduced to FeO and it in turn combines with silica to form faylite (2 FeO. SiO_2). Faylite melts at 1290°C in the bed and thus wets the solid particles resulting in their bonding into a big and strong agglomerate. Most of the heat generated out of combustion is consumed in drying, preheating and calcination of materials in the lower layer of bed. When the combustion zone reaches the bottom layers of the mix, outgoing combustion gases attain maximum temperature indicating that the sintering of the mix has been duly completed. The sinter cakes discharged from the grate are broken by crushers to suitable size, then screened and oversize is cooled and stored or despatched to blast furnaces. The undersize is recycled back for sintering. The top size of iron ore fines should not exceed 10 mm and should be preferably below 6 mm with not more than 20 % -100 mesh material. Excessive quantity of very fine material affects the sinter bed permeability, therby causing high pressure drop during drawing of air through the bed and this phenomenon affects sinter output and increases fuel consumption. The return fines in the sinter should be less than 25%. Excess quantity of return fines reduces porosity of sinter. Moisture content in the mix should be between 7 and -12%. The quantity of coke breeze used is 6 to 8% of the ore. The quantity of flux depends upon the gangue content of the ore and usually proportioning of the flux material is regulated to obtain self-fluxed or super-fluxed sinter ($CaO/SiO_2 > 2.0$). The flux addition in the sinter mix improves the physical quality of sinter. It also reduces the iron content of blast furnace slag due to presence of iron in sinter in more reducible form (oxides and ferrites). Use of self-fluxing sinter in blast eliminates the need to include raw limestone in the burden, an undesirable constituent in respect of thermal efficiency (decomposition of limestone, being endothermic in nature, consumes lot of heat).

Use of super-fluxed sinter markedly improves its reducibility without affecting its strength. This is due to the presence of easily reducible calcium ferrites in large quantity instead of less reducible ferrous silicate (faylite).

Sinters are porous and brittle and therefore they cannot be transported over a long distance as they cannot withstand the rigors of handling, i.e. repeated loading, unloading, etc. That is why sinter plants are located close to the iron blast furnaces.

Before oil crisis, more emphsis was laid on producing sinter with better cold strength (SI : shatter index) and less degradation during reduction when the blast furnaces were operated with high ore/coke ratio (less coke) and high oil injection. Now the scenario has changed and blast furnaces are being operated with high coke and low oil injection. This has led to the production of highly reducible sinter with better high temperature properties. Sintering with low heat input is considered the best method to achieve the objectives in the changed scenario, i.e. better reducibility, less slag volume, good high temperature properties (softening, melting and dropping characteristics) and optimum SI and RDI. The following developments in sintering have played a significant role in achieving the objectives:

i) **Use of fine quartz and serpentine**[2,6] : When fine quartz and serpentine (-1 mm size) are incorporated in the sinter mix, slag formation rate speeds up and productivity and strength (SI) of sinter are improved and slag volume in the blast furnace is reduced.

ii) **Addition of quick lime in the sinter mix**[2,6] Quick lime addition in sinter mix plays a great role in production of sinter with increasing calcium ferrite concentration with improved LTB resistance. Fine lime particles fuse and coat nucleus (1-5mm) and medium size particles (0.25mm to 1mm) to form pseudo- particles. Due to addition of quick lime, the amount of pseudoparticles increases up to 30% from as low as 2% in the feed.

iii) **Deep bed sintering** : Increase in bed depth from 350-400 mm to 600-650 mm has led to improvement in sinter productivity rate with low heat input. Nippon Steel Sintering operation and Tabato Sinter Plants in Japan, by increasing bed depth by 125-150 mm, have been able to produce sinter of better quality with decrease in coke breeze requirement by 5-10 kg/t of sinter. TISCO (India) also got similar results.

iv) **Double layer sintering** : The world biggest sintering machine (effective grate area : 600 m^2) at Wakamatsu (Japan) employs double layer sintering process, i.e. to increase the amount of coke in the upper layer of bed, which otherwise gives yield of sinter due to lack of heat. This system also decreases coke content in the lower layer which otherwise would remain for long time at high temperature due to excess heat. Duble layer sintering ensures good bed permeability and uniform sintering reaction throughout the bed. This has helped in producing sinter of good quality and high productivity with less fuel (coke breeze less by 4kg/t) and reduced air (0.5 Nm^3/t) consumption. M/s Wakamatsu is operating with high sinter bed depth (630 mm).

v) **Improved ignition conditions**[2,6] : Sintering process is initiated by ignition of coke breeze on or near sinter mix surface at about 250-650°C. Lack of oxygen in the fuel gas delays this ignition. A rapid ignition is necessary for optimum utilization of sinter strand area and it is possible only with full combustion of coke. Increase in length of furnace hood by 10% of the strand, use of coke oven gas, and oxygen-rich fuel (15-18%O_2) reduce fuel consumption, improve sinter productivity and increase shatter strength.

vi) **Heat recovery from exhaust gases**[2,6] : The sintering process accounts for about 10% energy consumption of an integrated iron and steel plant. About 50% of the heat required for sintering is discharged into atmosphere in form of waste gases (30% from hot air of sinter cooler +20% from waste gas). Kokura Steel work of Sumitomo Metal Industries, Japan has developed a technology to recover heat from waste gas by sending them to heat exchangers and hot air from exchanger is utilized to increase

sintering temperature. The exhaust air from cooler utilized is preheating air for coke oven gas and also to enrich its oxygen content.

vii) Step-by-step box sintering process[8]: Small DL sintering machines are prone to the problems of excessive air leakage, high energy consumption and poor quality sinter production. The step-by-step box sintering process developed in China provides solution to these problems. In this process, ore fines are sintered in a box. The sintering takes place in four stages. A loaded box is ignited and sintered at ignition sinter position (ISP), then driven to cooling position, then moderate cooling position (MCP) and finally to discharging stage and thereafter ready for next loading. The ISP box is pushed forward to CP, and another loaded box occupies the ICP instantly. In this way, continuous sinter production goes on. This method of sintering reduces energy consumption. In addition, capital cost is only 50% of the conventional DL machine and high quality sinter is produced. It is mainly suitable for low capacity sinter plants. Such type of sintering plants are already in operation in Sichuan, China.

10.1.3.3 Pelletization

The pelietization process is the formation of green balls (12-15 mm) by rolling fine ground ore or concentrate (generally 60 to 65% -32.5 mesh size) with very small quantity of bentonite (as binder) and 8-10% moisture and hardening the green balls by heat treatment (drying, preheating and firing) under oxidizing conditions up to a temperature of 1250-1350°C. As a result, oxide bridging, grain growth and slag binding occur and sufficient pellet strength is developed. The pellets are then cooled in air. Pelletization thus produces agglomerates in highly oxidized state as opposed to sinters which contain 5-10% ferrous iron. The oxidation of magnetite, being exothermic in nature, magnetite requires less fuel for pellet induration (6-8 therms/t) against 10-12 and 12-14 therms/t for hematite and limonite, respectively.[1,3] The fuel requirement in pelletization is usually 50% of fuel required for sintering. Different steps involved in pelletisation are as follows:

a) Feed Preparation : In pelletising process, the production of good quality green balls assumes paramount importance for making good quality fired pellets. The major bonding forces in balling are those due to surface tension of water and mechanical interlocking of particles on rolling. Any impurity present in the ore or hydrophobic flotation reagents which reduce the surface tension of water pose problems in balling. The moisture content is also critical for the production of good quality green balls. Hematite, magnetite and limonite require about 8% , 9% and 13% moisture, respectively. It is a known fact that feed for balling should be finer than -65 mesh size containing at least 65% material finer than 325 mesh size. Specular hematite grains require even fine grinding sometimes up to 100% -325 mesh. In balling process, it is desirable to have minimum porosity in the green balls (25-30%).

b) Balling : Balling normally starts when fine ground material mixed with binder is moistened and balled by rolling. First, small pellets (seeds) are formed. These seeds then grow layer by layer and get compacted by rolling action which reduces the size of the balls. Drums, discs and cones of different designs are used for balling. Drums are claimed to be the best for magnetite fines of beneficiated concentrates obtained by flotation/magnetic separation. Drums of 3 m dia with appropriate length (2.5 to 3 times the dia) are usually employed for balling. The pellets/balls produced being different in sizes need screening to separate undersize for recirculation back to the drums. In good operation, the recirculating load should not be more than 150%. It is to be noted that seed formation and growth of balls take place by cascading action and therefore it is necessary to have rough surface which is provided by 25-30 mm thick compact lining of ore inside the drum. The drum speed is 8 to 12 rpm. A single drum can produce up to 60 tph pellets. Addition of water by spray in drum tends to increase surging

and, therefore, it is advisable to operate four circuits simultaneously.

For natural fine ores like Indian blue dusts, disc pelletisers give better results. The present trend is to use disc of 6 m dia or larger even if the capacity of the plant does not warrant the use of larger discs as better quality pellets are obtained only with larger discs. The discs speed and inclination ranges from 5-15 rpm and 35 - 65°, respectively. The ore lining of 5-12 mm is provided in the discs for better cascading action. The retention time in the disc is usually 6-10 minutes. Discharge of pellets from discs takes place by centrifugal force when the pellets acquire a certain size. In case proportion of undersize pellets is not more than 5 to 8%, screening of discs pellets is not usually practised.

c)Hardening(Induration): Though hardening or Induration of green balls is often carried out in one equipment, it consists of four operations, namely, drying, preheating, firing and cooling.

In case of magnetite pellets, a recuperation period between firing and cooling is introduced to ensure completion of exothermic oxidation of magnetite to hematite.

Drying of pellets removes moisture and hardens the pellets to such an extent that they do not develop cracks in the subsequent operation. The pellets are dried slowly at 100-200°C to prevent rapid release of moisture causing cracking of pellets. The first stage of drying is always updraft. The down draught drying if done in the first stage, causes redeposition of moisture released from upper layer onto the lower layer and thus rendering them susceptible to deformation under load of pellet in the upper layer. After drying, the pellets are preheated grauduallly without subjecting them to thermal shock. In case limestone is used as binder, its calcination must be completed below 900°C to prevent bursting of pellets due to rapid release of carbon dioxide. Combined water as in case of limonite, goethite, etc. must be completely eliminated during preheating period.

Firing of preheated pellets is generally done just below the fusion temperature or below dissociation temperature of hematite. For acid pellets, the firing temperature is about 1350°C whereas for basic pellets (self-fluxing pellets), the firing temperature is maintained between 1250 and 1300°C. The induration cycle is adjusted in such a way that pellets of maximum strength are obtained with minimum fuel requirements.

The cooling of fired pellets is done to recover sensible heat so that the ensuing air can be used for preheating and firing. The induration of green pellet is generally done in any of the three furnaces, namely, vertical shaft, travelling grate and grate kiln.

Vertical Shaft Furnace : The shaft furnace was the earliest device used for pellet hardening. The shafts had an effective height of 15 m with 4.2m x 1.8m rectangular cross-section. The shaft furnace operates on the counter current flow principle, i.e. heat extracted from cooling of pellets is used for high temperature heating of pellets. Thermal efficiency of shaft furnace is high. Fuel requirement for firing of magnetite pellets in shaft furnace is about 400,000 - 500,000 Btu/t. capacity of shaft furnaces is small (1000-1200 tpd), a number of units are required for large capacity plants (more than one million tonnes/year)

Travelling (Straight Grate) Furnace : The green pellets are charged onto a travelling grate, broadly similar to sinter strand and subjected to drying, preheating, firing, and cooling as they travel along the strand. Grate bars and side walls of pellets are protected by covering with layers of fired pellets. Typical strands of 60-90 m length and 3 m width with about 400 mm pellet depth produce about 150-200 tph fired pellets. The induration cycle is flexible and drying, preheating and firing periods can be adjusted with ease to suit the needs of a particular material being pelletised. The heat is supplied by oil/gas burners in the hood. The hot air drawn from under firing section of the grate and also from above cooling section is used both to dry the green pellets and as combustion air for firing purpose. On a typical strand, 25% of the air is used for drying, 40% for preheating firing and 35% for cooling[7]. A new development is a continuous circular grate in which different

segments are used for different operations. It is claimed that this type of furnace is more suitable for very large output (5 million tonnes/year from a single unit)[10].

Grate Kiln Furnace : In this system, the green pellets are dried and preheated to about 1,000°C on a travelling grate and then fired in a rotary kiln and finally cooled in a separate cooler. The strand is shorter (30- 36 m) and simpler and does not require hearth and side layers of fired pellets for grate protection. Hot gases from the kiln are used first to preheat the dried pellets and then repassed through the strand to dry green pellets. The preheated pellets are charged into a rotary kiln inclined at certain angle. An oil/gas-fired burner is set at discharge end of the kiln and pellets travel down the kiln counter current to the combustion gases. The kilns are 36 m long with 4.5 m dia. The pellets from rotary kiln are discharged into an annual cooler (12 m dia with 0.8 m annular width). The air from the hotter end of cooler is used as secondary air for the rotary kiln burner. A single grate kiln furnace can produce 200 tph of fired pellet. The control of grate kiln plant is claimed to be easier as both grate and kiln can be controlled independently. It does not require high temperature fans. But the capital cost of grate kiln is high and it requires larger floor space.

10.1.3.4 Developments in Pelletisation [2]

The greatest challenge to pelletisation is to improve its performance in comparison to sinter in blast furnace opertion Eventhough pelletisation is superior to sintering in respect of thermal efficiency, heat input and product yield, its cost of production has increased substantially since oil crisis in 1978 because of its dependence on oil-derived fuels for firing. The large amount of electrical energy required for ore grinding and substantial increase in oil cost had made pellet production an uneconomical proposition. This led to the closure of some pelletisation plants in Australia, India and South America. In order to cope with these adverse eventualities, energy saving/reduction mesures, such as (i) decrease of moisture content in the green pellets, (ii) incorporation of char/coal in pellet mix, (iii) improvement in thermal efficiency at the grate by increasing the bed depth, (iv) use of pulverised coal partly or fully for pellet firing, etc. have brought down the cost of pellet production to the level of sinter production. When sinter and pellets are compared as a feed to blast furnace, it is claimed that pellets are inferior. Therefore, measures have been taken to improve the pellet performances. The melting and softening temperature of self-fluxed pellets, being lower, they are more prone to retardation of reduction phenomenon which occurs due to diffusion of FeO in the core and slag onto the metallic shells of pellets, resulting in clogging of pores. This problem has been tackled by adding MgO bearing minerals which raise the melting temperature of slag. M/s Pine Slazgitter Steel Plant (Germany) is producing hot metal employing 100% pellets burden (olivine pellets with 2-2.5% MgO). As an alternative measure, light weight or porous pellets have been produced by incorporating combustible material (saw dust, coke breeze, peat, moss, etc.) in the pellet mix which on firing burn out and leave pores behind

Because of its round shape and high bulk density, the repose angle of pellets is low and poses problem in maintaining a given burden distribution in blast furnace. This problem is tackled by crushing of large (Jumboo) pellets to the required size having angular shape similar to sinter.

The adoption of solid fuel firing technology (solid fuel is fed directly into combustion hood/kiln by means of suitably designed burners) dispenses with the use of costly petroleum products. Kakongawa grate kiln pelletisation plant at Kobe Steel (Japan) is producing pellets incorporating 0.8% coke breeze and dolomite in the green mix and is employing 100% coal firing technology. Fuel cost is reported to be reduced by 60%.

All these developments in pellet making have made pellets almost comparable to sinters in respect of production cost as well as performance in blast furnaces.

Cold Bonded Pellets : Pellets are usually heat hardened and fuel cost is the largest cost

ingredients in the pelletisation process. In order to reduce pelletisation cost, efforts have been made by many organisations to develop a pelletising process which requires little heat. Many cold- bonding processes have been developed.

A brief description of some of them is as follows :-

a) Gran Cold Pelletisation Process : This process was developed by M/s Grangesberg A.B. Sweden in 1960. It utilizes cement as binding material. Iron ore magnetite fines, ground cement and water, etc. are thoroughly mixed in mixer and then balled in a disc pelletiser. These 20-22 mm green balls are mixed with a portion of moist filtered concentrate and screened undersize of hardened balls in 2:1 ratio and this mix then goes to a hardening bin where it is hardened by curing for 30-40 hours during which it acquires 20-30% of its final strength. The fine concentrate and undersize pellets are added to prevent the balls from sticking together during hardening. The balls from this first hardening stage are screened and fines are recirculated and oversize pellets are crushed. Two additional curing stages lasting about 3 weeks are required to give final shipping strength (200 kg) to the pellets. The quantity of cement added is about 10% by weight and moisture content in the final hardened pellets is about 6%. Due to addition of cement, iron content in these pellets is low and carbonate in these pellets is in undecomposed form. Crushing strength is also low.

b) Carbonate Bonding (COBO) Process : In this process, ore fines are mixed with hydrated lime and coal fines (80% ore fines, 8% dolomitic monohydrate, 4% bituminous coal and 8% moisture) in a mixer and then subjected to balling. The green balls (10-15 mm) are dried in a rotary kiln to about 2% moisture with hot air (125 - 150°C). The dried balls are then treated in a second rotary kiln where CO_2 gas 10-30% by volume is admitted in the kiln at 50°C. Binding is caused by the reaction of CO_2 with monohydrate to form carbonates. This reaction is exothermic. The hardened pellets are screened.

c) Chor Process : This process is similar to Gran cold process but heat hardening is accelerated by steam curing.

d) Hydrothermal Process : Lime and silica which are used as binders dissolve to some degree under hydrothermal conditions and react to form calcium hydrosilicates which bind ore particles together. These pellets are equivalent to or even better than heat hardened pellets.

10.1.4 Direct Reduction

Direct reduction is the reduction of iron oxide in the feed stock (ore, pellet, sinter, etc.) by solid/gaseous reductants in solid state i.e. without melting any of the material involved. The product of a DR process is called direct reduced iron (DRI). It has the same physical form as before reduction and contains all the gangue minerals (silica, alumina, lime and magnesia) present in the original feed[11] [26]. DRI was referred also as sponge iron because of its porous character acquired as a result of reduction. Now-a-days, DRI produced by gas reduction is briquetted to prevent reoxidation and therefore porous character of the reduced material is eliminated. The DR contains mostly metallic iron (92-95%), a few percentage of FeO, iron carbide (Fe_3C) and impurities associated with feed stock (ore, pellets, coal, flux, etc.). A certain quantity of residual oxygen in form of FeO is required to ensure boiling action in steel making by Electric Arc Furnace (EAF). The typical reduction reactions in DRI production are as follows :

Reduction with carbon :

$$2\,Fe_2O_3 + 3C + heat = 4\,Fe + 3\,CO_2 \qquad (1)$$

$$Fe_2O_3 + C + heat = 2\,FeO + CO \qquad (2)$$

Reduction with Carbon monoxide :

$$Fe_2O_3 + 3CO = 2\,Fe + 3\,CO_2 + heat \qquad (3)$$

$$Fe_2O_3 + CO = 2\,FeO + CO_2 + heat \qquad (4)$$

Reduction with Hydrogen :

$$Fe_2O_3 + 3H_2 + heat = 2\,Fe + 3\,H_2O \qquad (5)$$

$$Fe_2O_3 + H_2 + heat = 2\,FeO + H_2O \qquad (6)$$

Reduction for Carbide Formation :

$$3\,Fe + CO + H_2 = Fe_3C + H_2O \qquad (7)$$

Based on solid and gaseous reductants, a number of DR processes have been developed

TABLE 10.1: DIFFERENT PROCESSES FOR DRI PRODUCTION

Process	Reductant	Bed condition	Type of furnace and cooler	Product
SL/RN	Coal	Moving	Rotary Kiln Rotary Cooler	Pre-reduced pellet or ore
Hoganas	Coal fines	Static	Staggers in tunnel kiln	Porous pre-reduced ore
Freeman	Carbon fines	Moving	Rotary Kiln	Pre-reduced pellets
Krupp Codir	Coal	Moving	Rotary Kiln Rotary Cooler	Pre-reduced pellets or ore,fines briquetted
DRC	Coal	Moving	Rotary Kiln Rotary Cooler	Pre-reduced ore or pellets
ACCAR	Coal	Moving	Rotary Kiln Rotary Cooler	Pre-reduced ore or pellets
Salem	Coal	Moving	Rotary Hearth Rotary Cooler	Pre-reduced ore or pellets
Midrex	$CO + H_2$	Moving	Shaft furnace	Pre-reduced ore or pellets
HYL	$CO + H_2$	Static	Fixed bed reactor	Pre-reduced ore or pellets
Armco	$CO + H_2$	Moving	Shaft furnace	Pre-reduced ore or pellets
Purofer	$CO + H_2$	Moving	Shaft furnace	Pre-reduced ore or pellets
Fior	$CO + H_2$	Fluid	Fluidized bed reactor	Pre-reduced ore fines
ACCAR	$CO + H_2$	Moving	Travelling grate kiln	Briquetted pre-reduced pellets
HIB	$CO + H_2$	Fluid	Fluidized bed reactor	Briquettes
H-Iron	H_2	Fluid	High pressure fluidizing bed	Sponge iron briquettes

and some of them have been listed in Table 10.1 and energy and power requirement for various processes are shown in Table 10.2

10.1.4.1 Brief Description of Some DR Processes

a) SL/RN Lurgichmieund Huttentechnic GmbH,Germany[2,8] : Almost all DR processes use rotary kiln when reductant employed is coal. The feed to SL/RN process consists of ore (5-10mm), pellets (8-15 mm) and sinter with 1.2 basicity, all with 65% iron (minimum), coal (-15 mm) with less than 1.5% sulphur, recycled char, flux (limestone, dolomite : 1 - 3 mm size) if sulphur needs to be scavenged from the coal. The rotary kiln is fed with charge up to 15-20% of its volume for optimum kiln volume utilization and for preventing reoxidation. The charge moves counter currently with oxidizing gas in the free board. In the preheating zone (40-50% kiln length maximum for better kiln efficiency), the charge is heated to 980°C. Reduction starts at about 900°C. The volatile component and excess carbon monoxide are burnt by the addition of air introduced along the kiln. The fixed carbon/Fe consumption ratio is 0.42 for pellets, lump ore and concentrate and 0.38 for mill scale. The reduced material is

TABLE 10.2: ENERGY REQUIREMENT IN DIRECT REDUCTION PROCESS

Process	Reductant	Fuel (Reductant) (10^6 Btu/NT)	Power (kWh/NT)	Total Energy 10^6 Btu/NT
SR/RN	Coal	12-23	75	17.8-23.8
Krupp Codir	Coal	13.8-19.3	50	14.3-19.8
DRC	Coal	19.4-19.9	110	20.5-21.1
ACCAR	Coal/Oil/Natural gas	16-16.6	50-75	16.5-17.4
Salem	Coal	18.9-26.0	35	19.3-26.4
Midrex	Natural gas	10-11.7	100-115	11-12.8
HYL	Natural gas	10.2-11.0	100	11-11.2
Armco	Natural gas	11.2-11.8	30-35	11.5-12.0
Fior	Natural gas	15.1	57	15.7
Purofe.	Natural gas	12.0	118	13.2

discharged from the kiln via a transfer chute into a sealed rotary cooler. Water spray on the cooler shell reudces the temperature of the material to about 95°C in non-oxidizing atmosphere. Cooler discharge materials are continuously separated into DRI, DRI fines, and non-magnetic by means of screening and magnetic separation. Char is separated from the waste by gravity separation. The SL/RN DRI has a denser surface due to higher operating temperature as compared to gas reduction and also due to trundling motion in the kiln and hence it has less tendency to reoxidation dispensing with the expensive briquetting.

b) Midrex Process (Midrex Corporation, a Korf Group Company)[11] : The main components of the plant are the DR shaft furnace, gas reformer and cooling gas system. The burden flows continuously into top of the furnace through seal legs and moves by gravity to preheating reduction and cooling zone. The cooled DRI is continuously discharged through seal legs at the bottom of the furnace. On discharge from the shaft, DRI is screened for removal of fines. Spontaneous ignition and reoxidation of DRI fines are taken care of by Midrex Chemaire Passivation Process. The reducing gas (95% H_2 + CO) enters the furnace through bustle pipe and ports located at the bottom of reduction zone and flows counter current to descending charge. Reduction takes place at about 900°C (760- 930°C). The partially used reducing (top) gas (70% H_2 + CO) flows into a top gas scrubber for cooling and dedusting. The largest portion of this gas is enriched and recompressed with natural gas and preheated (400°C) and piped into the reformer tubes for reforming to about 95% CO + H_2 and is finally recyled at 900°C to the DR furnace. The rest of the top gas provides fuel for the burners in the reformer. The reforming reaction is $CH_4 + H_2O - CO + 3H_2$, $CH_4 + CO_2 = 2\,CO + 2H_2$.

c) The Fior Process (Davy-Mckee Corporation) : Fluidized iron ore reduction (Fior) process reduces iron ore fines in series of four fluid bed reactors. Fresh reducing gases are produced by reformer and reduced iron ore fines are passivated by briquetting. The ore fines are fed continuously into preheating reactor where moisture is driven off and ore is heated to 800°C. The combustion products are used to heat and fluidize the ore. The preheated ore overflows into first of the three reduction reactors where 10% reduction takes place at about 700°C and thereafter ore overflows into other reactor at 750°C to yield reduced fines with 91-93% metallisation. The reduced fines are discharged from the final reactor into the briquetting feed bin from which they are fed (in hot condition) to the briquetting press (briquetting roll) to obtain pillow-shaped briquettes which pass on to trommels and then cooled in rotary cooler. A thin film of iron oxide forms on the surface of briquettes rendering them inert.

10.1.4.2 New Technology: DR processes

a) SKF Plasmasmelt Process : In this process, the 50% prereduced ore is injected together with coal and flux into the hearth of reactor. Energy to the hearth (1200 kWh/T) is supplied by

plasmatorch. The principal reductant in the shaft is coal. The hot gases ($CO+H_2$) form the shaft pass through two-stage fluid beds which produce 50% prereduced iron. The off gas from the shaft (1000 - 1200°C) is cleaned and cooled to 650°C before entering the two-stage fluid bed. The exit gas (30% CO + H_2)from fluid bed is used for drying and preheating the ore. The hot metal and slag are tapped intermittently as in the blast furnace. The liquid iron produced is qualitywise similar to hot metal from blast furnace and therefore can be refined to steel in conventional BOP operation.

b) Elred Process (ASEA/Stora, Sweden) : The Elred process comprises the pre-reduction of fine grained iron ore concentrates (size less than 150 mesh) with pulverised coal (0.2 to 0.3 mm) in a fluidized bed reactor followed by final reduction and smelting to liquid iron by plasma under the electrode in DC Arc furnace. Dry iron ore is fed pneumatically into venturi preheaters where feed is heated to 700°C by off-gas from the fluid bed reactor. After cycloning, the material enters fluid bed reactor through its base. Combustion air and dried coal fines are injected into the main section of the fluid bed operating at 7 atmospheric pressure and 950-1000°C temperature. One-third off-gas after cooling, cleaning, reheating and compressing is reused for fluidization. The pre-reduced material (65% metallisation) from reactor is transferred to DC arc furnace at about 600-700°C for melting, carbonisation and final reduction. The hot metal produced contains 3.5%C, 0.05% Si and 0.05% Mn and 0.3% S; FeO content of the slag is about 10%. For producing steel by BOP, the hot metal needs to be desulphurised.

c) Inred Process (Boliden Kemi Ab, Sweden) The Inred Process is the method in which iron ore concentrates are reduced to molten iron in two stages in a single flash smelt reactor. In first stage flash smelting of the concentrate by coal and oxygen is accomplished. The ore is prereduced to FeO and charge is superheated. About 90% of the process energy is supplied In this stage. Coal partly burns and the rest forms coke. In the second stage, prereduced heated material and coke react together to produce hot metal quality-wise similar to blast furnace hot metal. The temperature in upper and lower portions of flash smelt reactors is 1900 and 1450°C, respectively.

Scope for direct reduction : Very limited reserves of coking coal required for conventional iron and steel making route (BF-BOP route) and abundance of noncoking coal in the world have cleared the deck for alternative non-conventional routes to take over. DR-EAF (Direct Reduction and Electric Arc Furnace) route of iron and steel making is one of them. Major countries are producing DRI based on gaseous reduction. However, the resources of natural gas, being very limited, we have to fall back upon plentiful resources of low grade coal. DRI in general contains 0.2 to 0.5%C which is not sufficient for elimination of impurities and deoxidation of FeO. The feed to EAF requires 0.3 to 0.4% more carbon which DRI is unable to supply and therefore carbon containing scraps are added[26]. Supply of scrap at competitive cost being out of question, DRI needs to be recarburised. R&D studies are in progress in many countries and if successful results obtained are commercially viable, 100% DRI can be fed to EAF for steel making.

It goes without saying that most of the beneficiated concentrates end up in fine form as such are unsuitable for iron making without agglomeration. Adopting Inred, Elred, Plasmasmelt Processes, etc. which are capable of producing liquid iron qualitywise similar to blast furnace hot metal, utilizing fine beneficiated concentrate as such and pulverised coal may be a viable proposition in the coming years. Very serious consideration should be given to them after weighing pros and cons of the various emerging alternative.

10.1.5 Quality Evaluation of burden Material

Careful preparation of burden to provide burden of consistent chemical and size composition is imperative to ensure high productivity. Intensive work on quality improvement and burden preparation have led to the identification of certain properties which play vital role in furnace operation. Many testing methods have been and are being

developed all over the world to determine and quantify various properties. However, these testing methods are seldom comparable from one country to another and as a result there has been movement towards adoption of internationally recognised test procedures under the aegis of International Standards Organisation (ISO).

10.1.5.1 Testing of Feed Material for Blast Furnace

Testing Methods for Green Balls : Green balls cannot be stored during their transportation for balling to the induration unit due to their low mechanical strength. The plant managements therefore intend to keep transportation route as short as possible with a minimum number of transfer points and low drop heights. In modern travelling grate plants, there are four to five transfer points from 60 to 100 cm height.

a) Drop Number : The drop number indicates how often green balls can be dropped from a height of 46 cm before they develop cracks. Ten green balls are individually dropped onto a steel plate. The number of drops is determined for each ball before it cracks or crumbles. The average value of drops withstood by ten balls is taken as the drop number. According to experience, the minimum satisfactory value is 4. If the values are less, ball formation conditions are changed. Increase in binder quantity may increase drop number.

b) Crushing Strength : A certain minimum crushing strength is necessary for enabling green pellets to withstand compression load encountered in pellet bed on belt conveyor, drying grate, indurating grate, or in a shaft furnace. The average crushing strength of green and dry pellets is determined by compressing at least 10 pellets between parallel steel plates up to their breaking point. Average value of ten pellets is taken as crushing strength. The testing is carried out on a platform balance with weight indication by a pointer. The pellet to be tested is placed on the lower steel plate of the balance and is gradually compressed with a steel plate. The pellet breakage is indicated by the indicator. A crushing strength value of 1.8 to 2.3 kg is considered satisfactory.

10.1.5.2 Testing Methods for Burden Materials [12],[13]*: (Ore, Fired Pellets, Sinter and Briquettes)*

Determination of physical properties of ore and agglomerates is very important in getting information about how a material withstands transport from mine/production unit to iron making centres. Resistance to shatter as a result of dropping and resistance to abrasion and compression indicate the strength of burden material.

a) Shatter Strength : The shatter test, which was developed for coke is now employed for sinter and ores. A typical shatter is one in which 20 kg of ore or sinter greater than +10 mm in size is dropped four times from a height of 2 m. The number of drops for determining the shatter strength depends upon the number of drops the ore or sinter has to undergo during transport. The dropped material is screened over 10 mm screen. Percentage of +10 mm material present gives the shatter indices. Shatter indices in the range of 80-83% are indicative of strong burden material (sinter or ore).

b) Compression Strength (Crushing Strength)[7] : Compression or crushing strength is usually measured for pellets. Less attention has been paid for determining compression strength of other burden material (ore and sinter). In order to determine crushing strength, one individual pellet is placed between two plates in a similar manner as for green balls and compression load employing hand press/ hydraulic press/electrically operated press is applied a uniform time schedule. Pellets of an optimum roundness and most uniformed diameter are selected to ensure an even point load. A minimum of ten pellets is used for each test. In case of travelling grate plants, the test is carried out for the upper, lower and middle layers of pellets. The average strength of ten/thirty individual pellets is taken as the cold crushing/compression strength of pellets. The pellet should have at least crushing strength values of minimum 200 kg/pellet and 10%

pellets should not have crushing strength below 200 kg and 5% pellet strength below 50 kg.

The L.K.A.B. Laboratories in Sweden have developed a test procedure for determining the crushing strength of burden material (ore, sinters, etc.). About 2 kg of 10-15 mm material dried at 105°C is placed in a steel cylinder with an inner diameter of 200 mm closed with free running piston and having a removable base. The cylinder is then placed under a press and a pressure of 100 tonnes is applied. The pressed sample is screened over 5 mm screen. The percentage of material coarser than 5 mm is an index of compression strength. This index is claimed to be true measure of ability of pellets/ore lumps/sinter to withstand handling in transport (shipment). The values vary betwen 20% for very friable ore and 75% for very hard pellets.

c) Tumbler Resistance[7] : The shatter tests have been superseded by Tumbler Resistance tests (ASTM tests in USA and UK and MICUM tests in Europe). The basic features of these tests procedures are given in Table 10.3

The results of tumbling and abrasion tests can be readily correlated with shatter test results as per the following relationship:

+10 mm shatter index = 39.082 + 0.57 (ASTM + 6.3 mm Index). Pellets should have tumbler index 94% minimum and abrasion index of 5% maximum.

It is well known that pellets are high quality product in respect of physical strength. Ores, though usually more resistant to degradation in size than sinter can produce higher level of fines than pellets or sinters during handling. Friable ores are shipped without screening, i.e. with fines. Fines act as cushion and prevent further size degradation during transportation. This has led to the practice of screening burden material before feeding to the iron making furnaces.

10.1.5.3 Metallurgical Properties

It is a well known fact that some iron-bearing materials break down to a marked degree when heated under reducing conditions and other material tend to swell. Both these phenomena impair furnace permeability and in turn decrease production rate. Porous, open textured material with iron in oxide/ferrite form has good reducibility whereas dense lumps or slaggy material with iron present as silicate are difficult to reduce. For efficient operation of blast and other furnaces with high production and less heat input, ore lumps and agglomerate should have optimum metallurgical properties and it is necessary to have first-hand information about the behavior /performance of materials inside iron making furnaces. The following test procedures have been developed simulating

TABLE 10.3 : BASIC FEATURES OF TUMBLING AND ABRASION TESTS

Feature	ASTM	Micum	Half Micum	ISO	BIS (India)
Size (mm) (a) Ore Lumps or Sinter (b) Pellet	-50 +10 -38 +6.3	-40 +10 -25 +10	-40 +10 -25 +10	-40 +10 -40 +10	-40 +10 -40 +10
Weight (kg)	11.3	50	25	15	15 ± 0.15
Drum dia (mm)	915	1000	1000	1000	1000
Drum Length(mm)	458	1000	500	500	500
Number lifters	2	4	4	2	2
Size of lifter(mm)	51	100	100	50	50
Speed of drum(rpm)	24 ± 1	25	25	25 ± 1	25
Number of revolutions	200	100	100	200	200
Size of product after test(mm)	+9.5 +6.3 -0.595 (30 mesh)	+10 +5 -25	+10 +5 -2.5	+6.3 -0.5(28 mesh)	6.3 -0.5(28 mesh)
Tumbler Index	+9.5	+10	+10	+6.3	+6.3
Abrasion Index	-0.595	-2.5	-2.5	-0.5	-0.5

identical reducing condition in an endeavor to obtain meaningful information:

a) Descripitation Tests: Descripititation is experienced predominantly with ores. It occurs due to rapid release of free and chemically combined moisture and variable thermal expansion along and across the planes of ore structures when ores charged to the blast furnace enter a zone inside the furnace having well above 400°C temperature (upper stack portion of blast furnace). Ores without any moisture and high porosity (more than 10%) are less susceptible to decripitation. Ore with high decripitation index increases flue dust losses and affects furnace operation.

Testing Procedure : About 500 g of undried ore (20-40 mm size) is charged in a furnace vessel previously heated to 400°C until center of charge attains this temperature and is then held there for 30 minutes under a flow of 5,000 litres/hour of nitrogen. The sample is then removed, cooled and screened and percentage of 0.5 mm and -5.6 mm +0.5 mm material is determined. This test is conducted three times and average value indicates the decripitation results.

b)Low Temperature Breakdown (LTB)/ Reduction Degradation Index (RDI) Determination Tests: The disintegration of iron ore lumps, and agglomerates at 400-600°C in weakly reducing condition in the blast furnace is a well recognised phenomenon. It is reported that as the reduction advances from hematite to magnetite to wustite, disintegration Increases due to formation of micro-cracks and in some cases due to carbon deposition. High reduction disintegration increases flue dust production, causes scaffolding, impedes gas distribution, raises fuel consumption, affects reduction rate and furnace productivity. It can be controlled to some extent by careful selection of ores, by use of certain additives, etc. However, in spite of various efforts, all sinters and most of pellets are susceptible to considerable disintegration. The following tests have been developed to investigate this phenomenon:

Static Method (ISO 4696): The purpose of this test is the determination of degradation conditions prevailing in upper part of blast furnace. This test is carried out in a 75 mm dia. vertical tube furnace. The charge to the furnace comprises 500 g of 10-12.5 mm size. The test is performed at 500°C ± 10°C for a period of one hour under weakly reducing atmosphere (20% CO, 20% CO_2 and 60% nitrogen). After reduction, the material is tumbled in a 130 mm dia x 200 mm length rotating cylinder. The cylinder is rotated at 30 rpm for 300 revolutions. The tumbled material is then screened over 6.3, 3.15 and 0.5 mm screens. The +6mm and +3.15 mm fractions are designated as disintegration strength indices and -0.5 mm fraction as disintegration abrasion index. This method is primarily suitable for testing of sinters.

Dynamic Method (ISO/DP 4697): The test is carried out on 500 gm of 10-12.5 mm material (lump ore, pellets, sinters) in a small rotating drum (130 mm dia x 200 m length) at 500 ± 10°C for one hour under weakly reducing condition (60% N_2, 20% CO, 20% CO_2). The drum is equipped with four lifters which subject the test material to severe machanical load. The reduced material is subjected to screen analysis as indicated in the Static test. The dynamic methods are primarily used for pellets and ore lumps.

Swelling Test (ISO/DP 4698): Pellets when subjected to reduction at 900°C undergo excessive swelling and pose problems in furnace operation. It is reported that (i) pellets with less than 20% swelling pose no problem, and (ii) pellets with more than 40% swelling causes unstable blast furnace operation. Therefore blast furnace burden should have more than 65% of pellets with less than 20-40% swelling. The pellets to be tested are placed in a container which allows unrestricted expansion. This container in turn is placed within 75mm dia vertical tube. The pellets (10-12.5 mm) are tested at a temperature of 900°C for one hour employing 30% CO and 70% N_2 gas. The volume of reduced pellet is measured by means of mercury volumenometer or any other method and compared with original volume of the pellets. When reduced pellets are subjected to compression, the results indicate hot compression, strength of pellets. This one test alone helps in determining swelling index, hot compression strength and degree of reduction.

TABLE 10.4 : TESTS FOR MELTING & SOFTENING BEHAVIOR

Parameter	Method				
	Vawata Nippon Steel Japan	Kobe Japau	Mefos Sweden	RWTH, Achen Germany	SGA Germany
Retort dia (mln)	70	80	100	60	75
Sample weight (g)	500	500	1300	400	500
Sample size (mm)	10	9.5 - 12.5	10-15	7-15	10-12.5
Heating rate (C/min)	10 up to 800°C 5 above 800°C	16.6	3.3	10 up to 900°C 4 above 900°C	10 up to 900°C 4 above 900°C
Maximum temperature (C)	1400	1100	1325	1600	1600
Gas composition (RG)	30% CO 70% N_2	30% CO 70% N_2	35% CO 65% N_2	30% CO 70% N_2	30% CO 70% N_2
Gas programme	N_2 up to 200°C RG above 200°C	RG above 200°C	RG above "00°C	RG above 300°C	RG above 300°C
Gas quantity (l/min)	20	15	600	33	33
Load test (N/m^2)	10 above 800°C	20	5 -	6-11	5-10
Test Results	P.H. f(T)	P.H. f(T)	P.H. f(T)	P.H.R. f(T)	P H.R. f(T)

Where P : Pressure-drop, H: Sample height (Shrinkage), T: Temperature, R: Reduction degree, RG : Reduction gas, L : Loao, f : Function of.

10.1.5.4 Determination of Reducibility

An important characteristic of burden material is its reducibility, i.e. ease with which oxygen can be removed. Reducibility data indicate the amount of fuel required for reducing a material of particular size composition. The rate controlling step in reduction is the reduction of wustite to iron at 900-1000°C. There are two internationally standardised test procedures for reducibility determination.

a) Method as per ISO 7215 : The test is conducted in a 75 mm dia vertical retort on a 500 g of 10-12.5 mm pellets or 19-22.4 mm sinters/ore lumps at 900 $\pm$ 10°C employing a reduction gas containing 70% N_2 and 30% CO for 180 minutes. The test results indicate the degree of reduction.

b) Method as per ISO 4695 : This test is also performed in 75mm vertical retort on 500 g of 10-12.5 mm material (pellet, sinter, ore lump) at 950°C $\pm$10°C employing a gas containing 60% N_2 and 40% CO for 240 minutes or until 65% reduction is achieved which generally occurs after 60 minutes. The test results are reported as $\frac{dR}{dt} = 40$.

c) Reduction Under Load (RUL) -ISO 7992 : The method is suitable for determining the stability of pellet and blast furnace ores at high temperature. This test is conducted in a 125 mm dia vertical retort on 1200 g of 10-12.5 mm material placed under a load of 0.5 kg/cm^2 applied by pressure ram employing gas containing 40% CO, 2% H_2 and 58% N_2 for 150 minutes at 1050 $\pm$ 10°C. The test provides results pertaining to reducibility, shrinkage and resistance to gas permeability. This latter parameter dP_{80} at 80% degree of reduction is a measure of the stability of the pellets or ore during reduction.

d) Determination of Melting and Softening Behavior : Numerous testing facilities are available in various parts of the world for examining the melting and softening behavior. The Studiengesellschaft fur Eisenerzaufbereitung (SGA) Liebenburg, (Germany) conducts these investigations in the so called REAS test in which, through a special temperature and gas programme, a blast furnace is simulated. In continuous tests both indirect reduction by gas and direct reduction by carbon are studied up to 1550°C for mixture of burden containing ore lumps, sinters and pellets as the case may be. High temperature properties like softening, melting and dropping of iron and slag phases are recorded and evaluated. The information pertaining to various tests procedure being adopted in number of countries is given in Table 10.4

The softening and melting properties of sinter at high temperature (high temperature properties) have significant effect on gas flow in lower portion of blast furnace on reduction of ore due to its contact with gas and on melting speed. Self-fluxed sinters neither soften nor shrink even above 1100°C whereas acid pellets shrink and soften at high rate around 1000°C.

As a result of evaluation of properties and quality of burden material, it is reported that for better blast furnace performance, the burden should possess the following qualities:

Size composition	:	85% of burden in 8-16 mm size range
Tumbler index	:	94% + 6.3 mm, 5% -0.5 mm (max)
Crushing strength	:	250 da N/P, 5% max 150 da N/P, 10% max 200 da N/P, 15% max
Differential Pressure (RUC)	:	15 mm WG at 80% degree of reduction
Reducibility at 1050°C	:	0.8%/min. rate of oxygen removal
Swelling	:	20% vol max (measured at 1000°C)

10.1.5.5 Testing of Feed Stock for Direct Reduction Furnaces [12]

Generally, ore lumps and pellets are used for production of DRI. The procedure being adopted for evaluating the quality of feed stock for DRI production is as follows. These procedures are based on temperature and gas composition prevalent in DRI production.

a) Reduction Degradation Index (RDI) : The Dynamic method adopted in case of blast furnace feed stock is also adopted in this case. Only the reduction temperature is different (750°C). Other conditions are almost same.

b) Reducibility & Metallisation : This test is carried out in a 75 mm dia vertical retort at about 800 - 850°C. Other conditions are as per ISO - 4695. Metallisation is studied by means of chemical analysis of Fe and Fe met content in the reduced material.

c) Sticking and Clustering Behavior : This test indicates the temperature at which sticking occurs. The test is conducted in a vertical retort (125 mm dia) as specified in ISO 7992. The sticking behavior is studied by screening over 20 mm screens and +20 mm fraction is evaluated by drop test.

For determining physical characteristics (compression strength and stability during transport), established methods are applied as is done in the testing of blast furnace feed stock.

10.2 IRON ORE TREATMENT IN INDIA

10.2.1. Characteristics of Iron Ore Deposits

India possesses vast resources of iron ore of which about 75% are hematite and the rest 25% magnetite deposits[14]. The hematite deposits comprise mainly hematite with minor amounts of hydrated iron oxides (limonite, goethite, etc.) and in some cases are intimately associated with gangue minerals like quartz, clay, felspars, mica, gibbsite, chlorite, etc. The magnetite ores are of low grade (30-40% Fe) containing quartz as the main gangue minerals and occurring mainly in Karnataka, Andhra Pradesh, Goa, Kerala and Tamil Nadu.

The hematite ores of Bihar and Orissa are in banded hematite quartzite formations and occur as massive ore (58 - 66% Fe, 1 - 4% SiO_2 and 1 - 6.5% Al_2O_3), blue dusts (extremely friable and flaky hematite powder with 66-68% Fe, 1-1.5% SiO_2, 1-2% Al_2O_3) and weathered lateritic ores (56-60% Fe, 1.7-3.8% SiO_2 and 5-7% Al_2O_3). Dalli-Rajhara deposit of Madhya Pradesh is high grade (64 - 65% Fe, 2-3% SiO_2 and 2-2.5 % Al_2O_3). Bailadila deposit is also high grade (66-68% Fe, 1-1.5% SiO_2 and 1.7-2% Al_2O_3) as well as hard in nature. The iron ore deposits in Bababudan hills of Karnataka have patches of thick bands and lenses of hematite in banded hematite and quartzite formations (57-62% Fe) whereas Donimalai deposits of Sandur have long narrow and scattered patches of hematite with intervening shale bands containing 58-64% Fe, 1-14% SiO_2 and 1.8 to 2.5% Al_2O_3. About 80% of iron ore production from Goa deposit is mainly from blue dusts (63-64% Fe, 2-2.8% SiO_2

and 3% Al_2O_3) and fines (59% Fe, 2.7% SiO_2O_3) and remaining 20% from the lumps (57-58% Fe, 3% SiO_2 and 7-8% Al_2O_3).

10.2.2 Status of Iron Ore Processing

Unlike iron ores in other countries, the Indian iron ore hematite deposits, though high in iron content, are characterised by high alumina and low silica content, i.e. high alumina silica ratio (1.5 to 3.0 for lumpy ore and 3-4 for fine ore[14]). Besides, the Indian hematite deposits are comparatively soft in nature. Therefore, it is imperative to prepare and beneficiate iron ores from the various deposits to obtain lumpy product (10-40 mm) for direct use in blast furnace and fine products for use in blast furnace after agglomeration by sintering and pelletisation.

The methods followed for processing iron ores in India comprise simple crushing and screening, crushing, scrubbing, washing, screening and hydroclassification and gravity separation. Multistage crushing, screening, hydroclassification/scrubbing are adopted in most of Indian iron ore processing plants based on hematitic deposits. The process adopted for treating magnetitic deposit at Kudremukh (Karnataka) comprises crushing, autogenous grinding, two-stage low intensity wet magnetic separation and gravity separation (three-stage spiralling). The flowsheets followed by of several Indian iron ore beneficiation plants are given in Figs. 10.1 to 10.8.

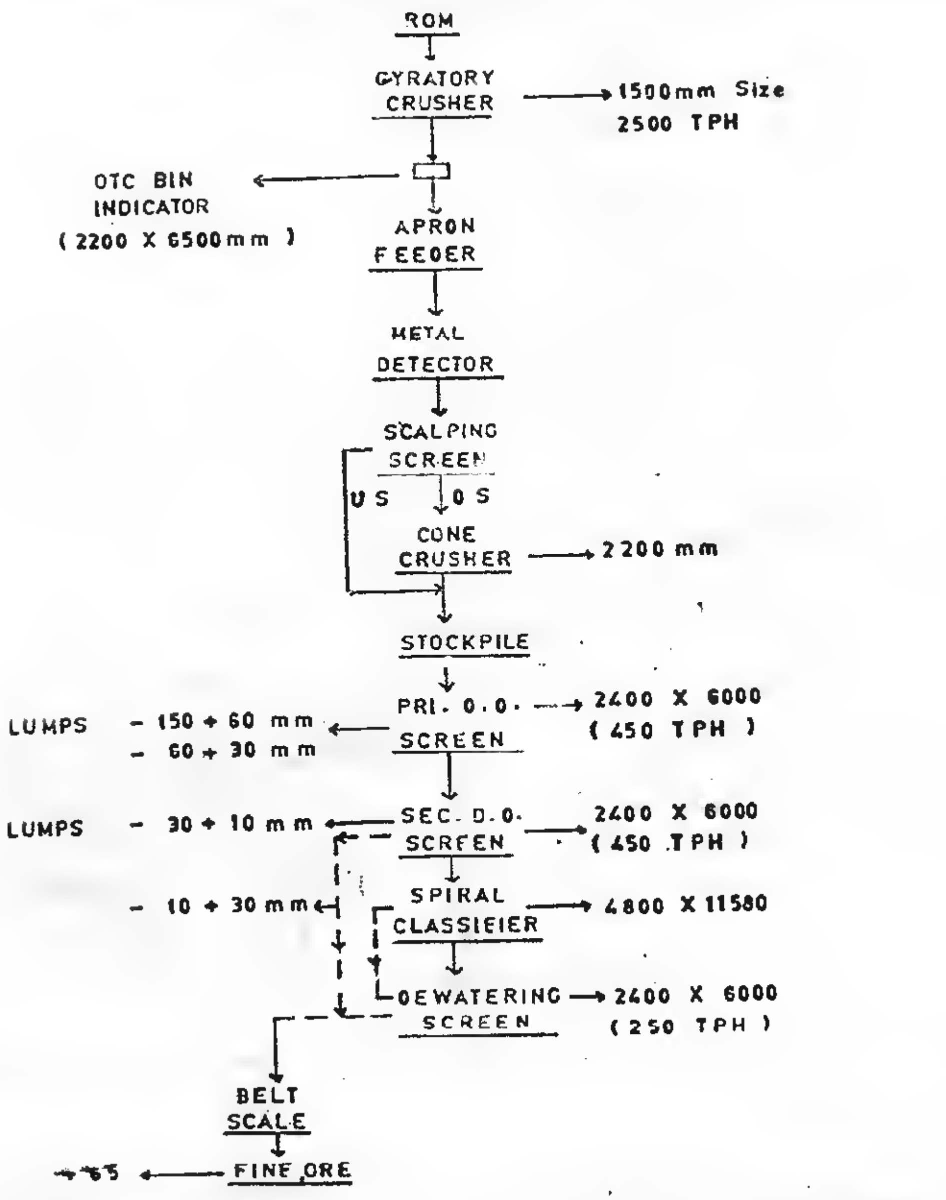
The information about crushing and other beneficiation equipment used in major iron ore processing plants is furnished in Table 10.5 and 10.6 , respectively.

TABLE 10.5 : CRUSHING EQUIPMENT IN MAJOR INDIAN IRON ORE PROCESSING PLANTS

Sr. No.	Plant Location	ROM Size (mm)	Product Size (mm)	Crusher size (mm) & Type			Stage of Crushing
				Primary	Secondary	Tertiary and Quarternery	
1	Barsua	1200	80	1524x2134 (Jaw)	610 (Red Gyratory)	--	Two-stages open circuit.
2	Bolani	800	50	915x1219 (Jaw)	432 (Red Gyratory)	1321 (Standard Cone)	Two-stage in open circuit, third one in closed circuit.
3	Bailadila-5	1200	100	1500 (Gyratory)	2200 (Standard Cone)	--	Open circuit
4	Bailadila-14	1200	100	1372 (Gyratory)	610 (Red Gyratory)	--	Open circuit
5	Daitari	800	100	1067 (Gyratory)	1200x1800 (Roll)	--	Open circuit
6	Dalli	1200	140	1500x2100 (Jaw)	2200 (Standard Cone)	--	Open circuit
7	Donimalai	1200	30	1372 (Gyratory)	2134 (Standard Cone)	2134 (Short Head Cone)	Two-stage in open circuit. Third in closed circuit.
8	Gua	1200	75	1524x2134 (Jaw)	2134 (Standard Cone)	--	Open circuit.
9	Kiriburu	1200	40	1524x2134 (Jaw)	610 (Red Gyratory)	2200 (Standard Cone) 2200 (Short Head Cone)	Three stages in open circuit. Fourth in closed circuit.
10	Kudremukh	1200	175	1524 (Gyratory)	--	--	Open circuit.
11	Meghataburu	1200	50	1372 (Gyratory)	2200 (Standard Cone)	--	Open circuit.
12	Noamundi	1200	50	1372 (Gyratory)	2134 (Standard Cone)	--	Second in closed circuit.
13	Rajhara	1200	100	1500x2100 (Jaw)	2200 (Standard Cone)	--	Open circuit.

Fig. 20.1 : Flowsheet of Bailadila Iron Ore Project of N M D C Ltd. in Dist. Baster (M.P.)

DEPOSIT. NO. 5 — CRUSHING/SCREENING PLANT — CAPACITY — 2500 TPH



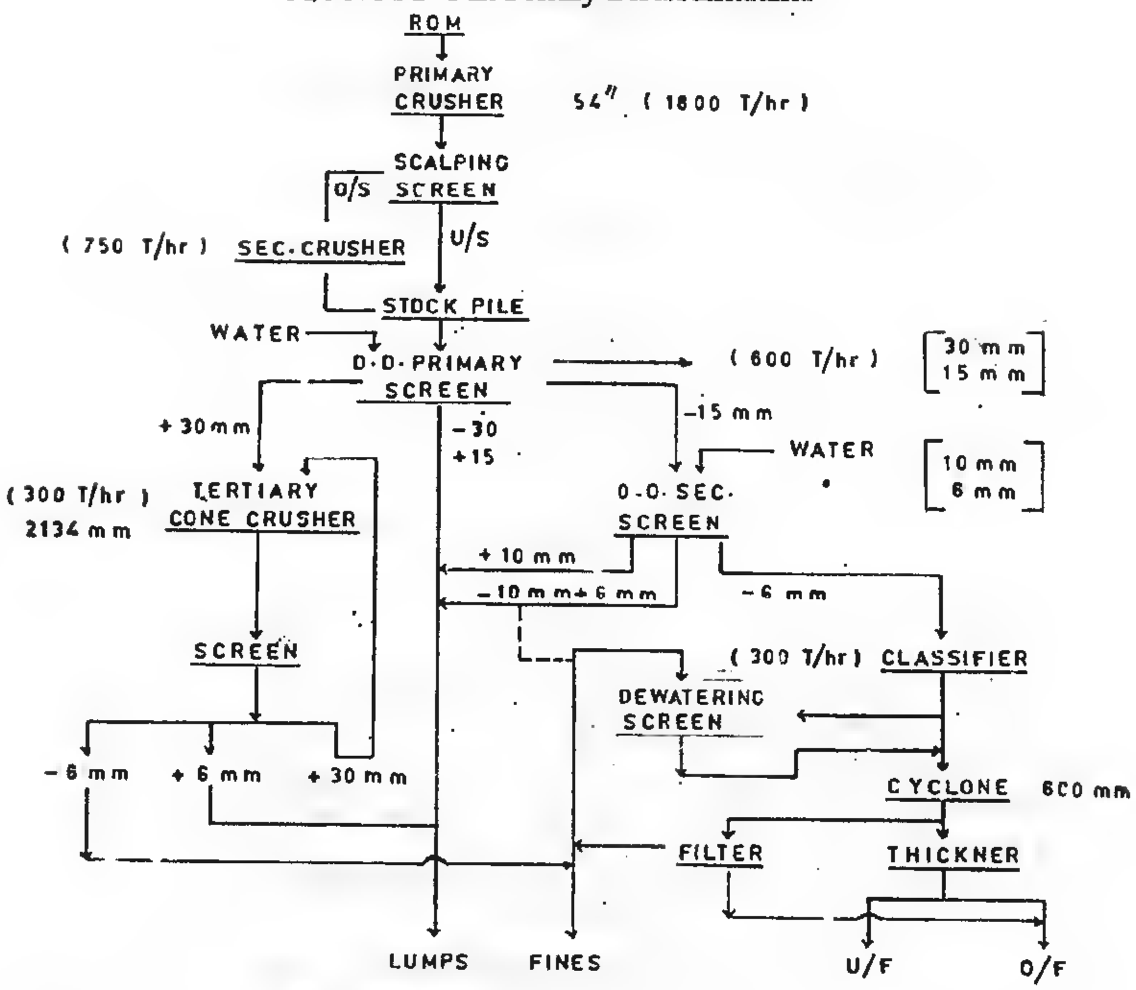
1 LOCATION — The plant is located at Bailadilla in the tribal region in Dantewara of Bastar dist. M.P.

2 CHEMICAL ANALYSIS ——

PRODUCT	% Fe	% Dist.
Feed	67.5 — 68.7	100.00
Lumps	67.5 — 68.3	—
Fine	65.2 — 69.0	—
Slimes	67.0 — 68.8	10—15

PROCESS — Crushing, Screening, Classification.

Fig. 10.2 : Flowsheet of Donimalai Iron Ore Mine of M/s N M D C in Bellary Dist..Karnataka



1 LOCATION — It is located in Bellary Hospet region, Sandur taluk, Bellary dist. Karnataka.

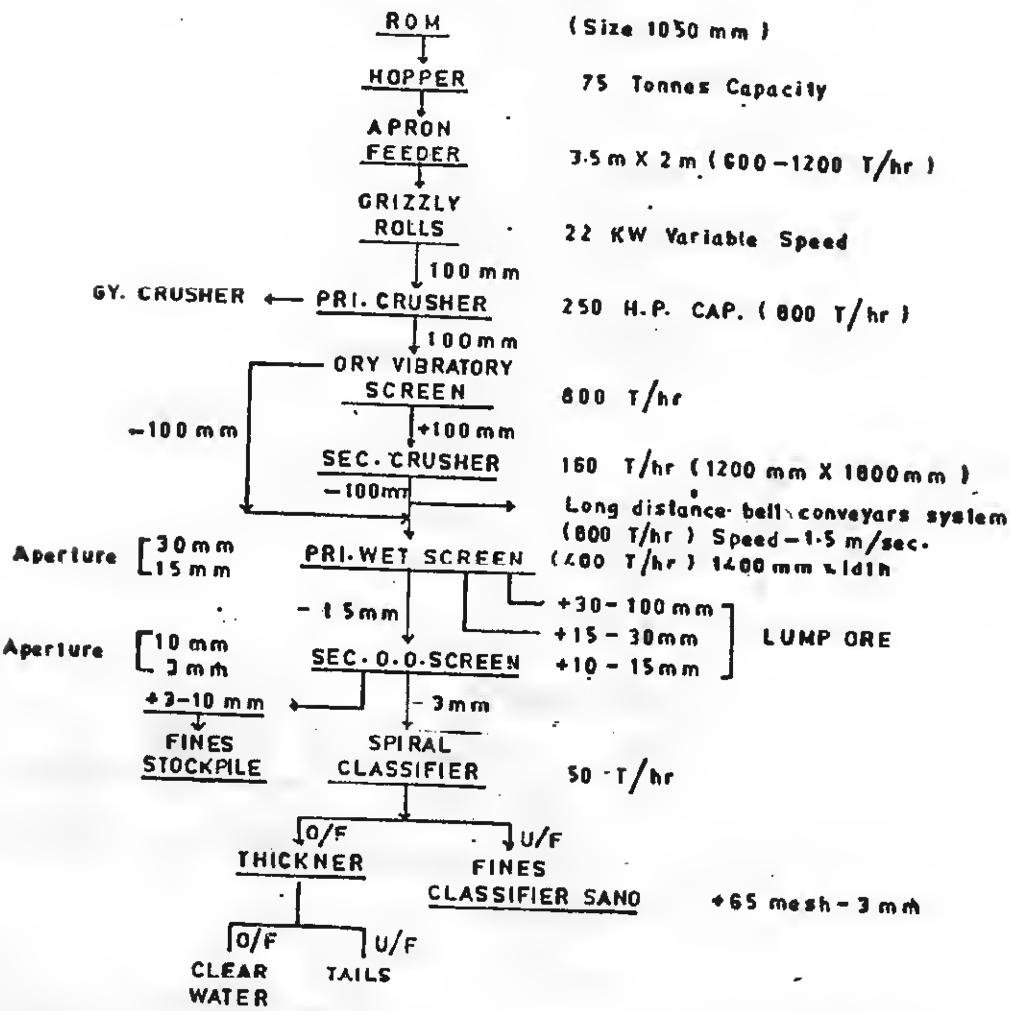
2 CHEMICAL ANALYSIS —

PRODUCT	% Fe	% Dist.	% SiO_2	% Al_2O_3	% P
Feed	64.40	100.00	3.18	3.01	0.07
Lumps	64.50	86.00	1.96	2.90	0.08
Fines	64.20		3.15	3.30	0.07
Tailings	60.10	14.00	6.7	5.30	0.09

3 PROCESS — Crushing, Washing, Screening, Classification, Cycloning.

Fig. 10.3 : Flowsheet of Daitari Iron Ore Project of M/s Orissa Mining Corpn. Ltd. in Keonjhar Dist., Orissa

PLANT CAPACITY — 800 T/hr



1 LOCATION — The plant is situated at the southern part of Talpada village, Keonjhar dist., Orissa.

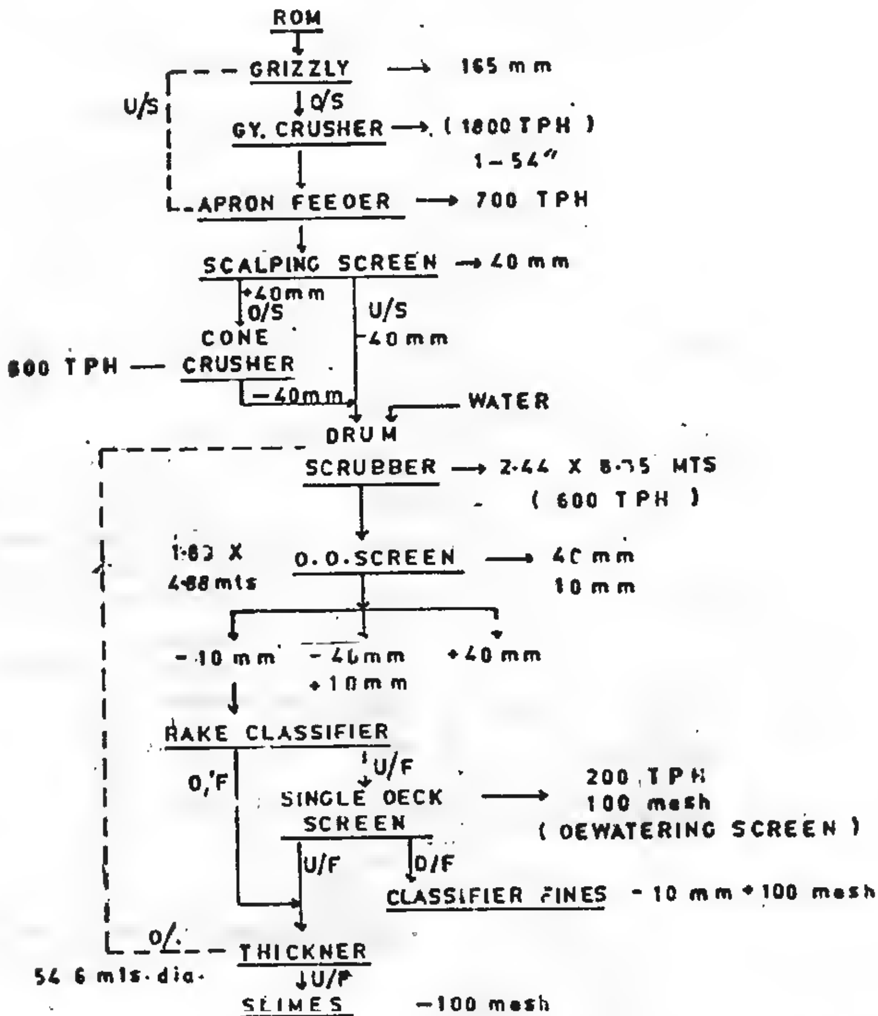
2 CHEMICAL ANALYSIS —

PRODUCT	% Fe	% Dist.	% Al_2O_3	% SiO_2
ROM	62.04	100.00	3.94	1.67
LUMP CONC.				
– 100 + 30 mm	63.52		3.57	1.06
– 30 + 15 mm	64.21	(45 – 50)	2.65	0.68
– 15 + 10 mm	61.51	90 – 95	3.12	1.11
FINES CONC.				
– 10 + 3 mm	67.54		3.32	1.01
– 3 mm	60.94	(40 – 45)	4.03	1.30
TAILINGS	49.10	8.66 – 11.80	12.98	7.31

3 PROCESS — Crushing, Screening, Classification.

Fig. 10.4 : Flowsheet of Noamundi Iron Ore Washing Plant of M/s TISCO Ltd. Singhbhum Dist., Bihar.

PLANT CAPACITY — 3.5 M.T./ANNUM
WET CIRCUIT — 2.6 M.T./ANNUM
DRY CIRCUIT — 0.9 M.T./ANNUM



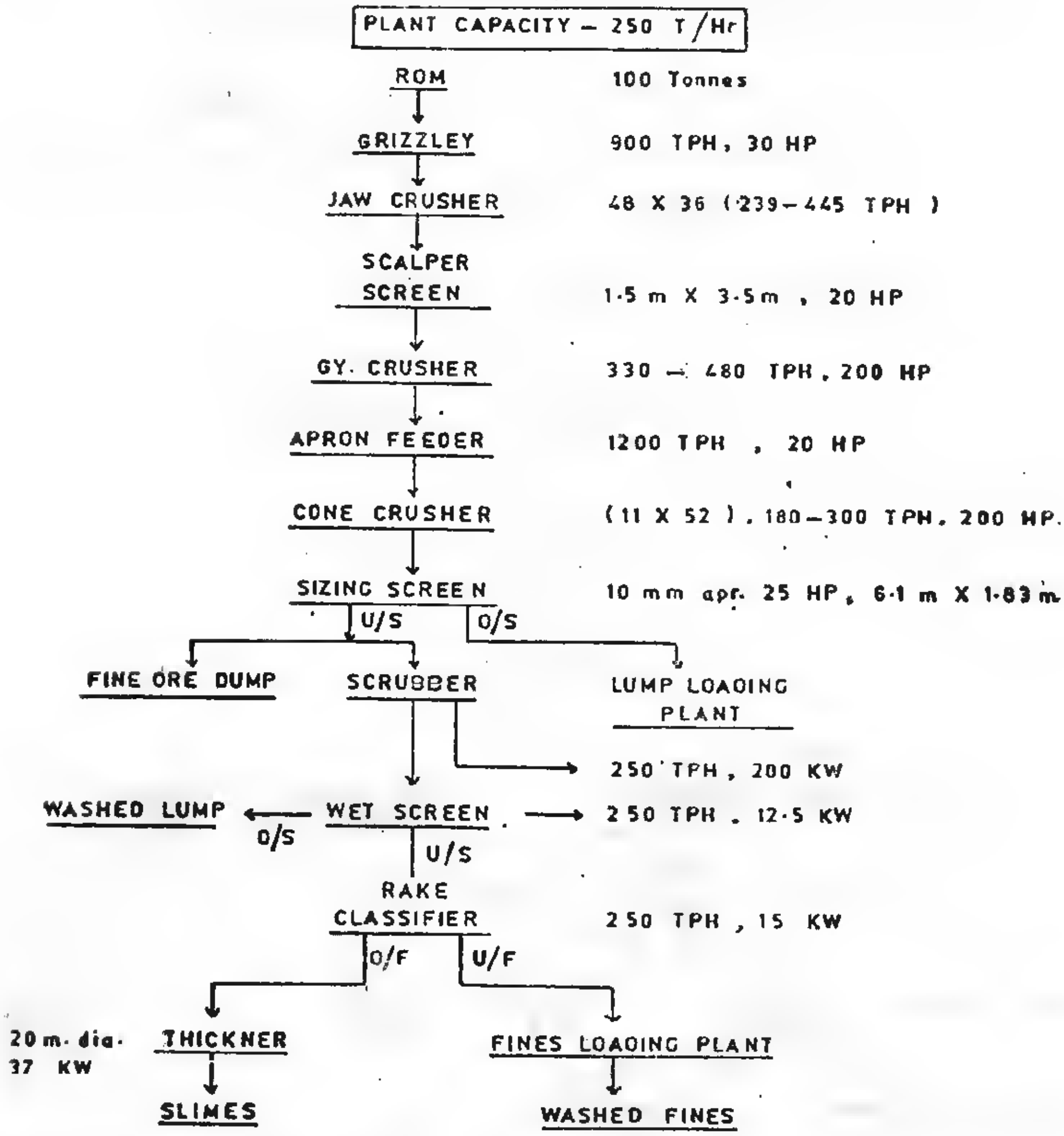
1 LOCATION — The plant is situated near Talanagar, Singhbhum, Dist. Bihar.

2 CHEMICAL ANALYSIS —

PRODUCTS	% Fe	% Dist.	% SiO_2	% Al_2O_3
Feed (ROM)	64–64.5	100.00	—	3–3.5
Sized Ore (−40mm + 10mm)	66.80	52.00	0.80	1.85
Classifier Fines (−10mm+10mm)	63.[illegible]7	34.00	1.20	3.65
Slimes (−100 mesh)	57.94	14.00	3.88	7.73

3 PROCESS — Crushing (Dry & Wet), Scrubbing, Washing, Screening, Classification.

Fig. 10.5 : Flowsheet of Bolani Iron Ore Fines Washing Plant of M/s Steel Authority of India Ltd.



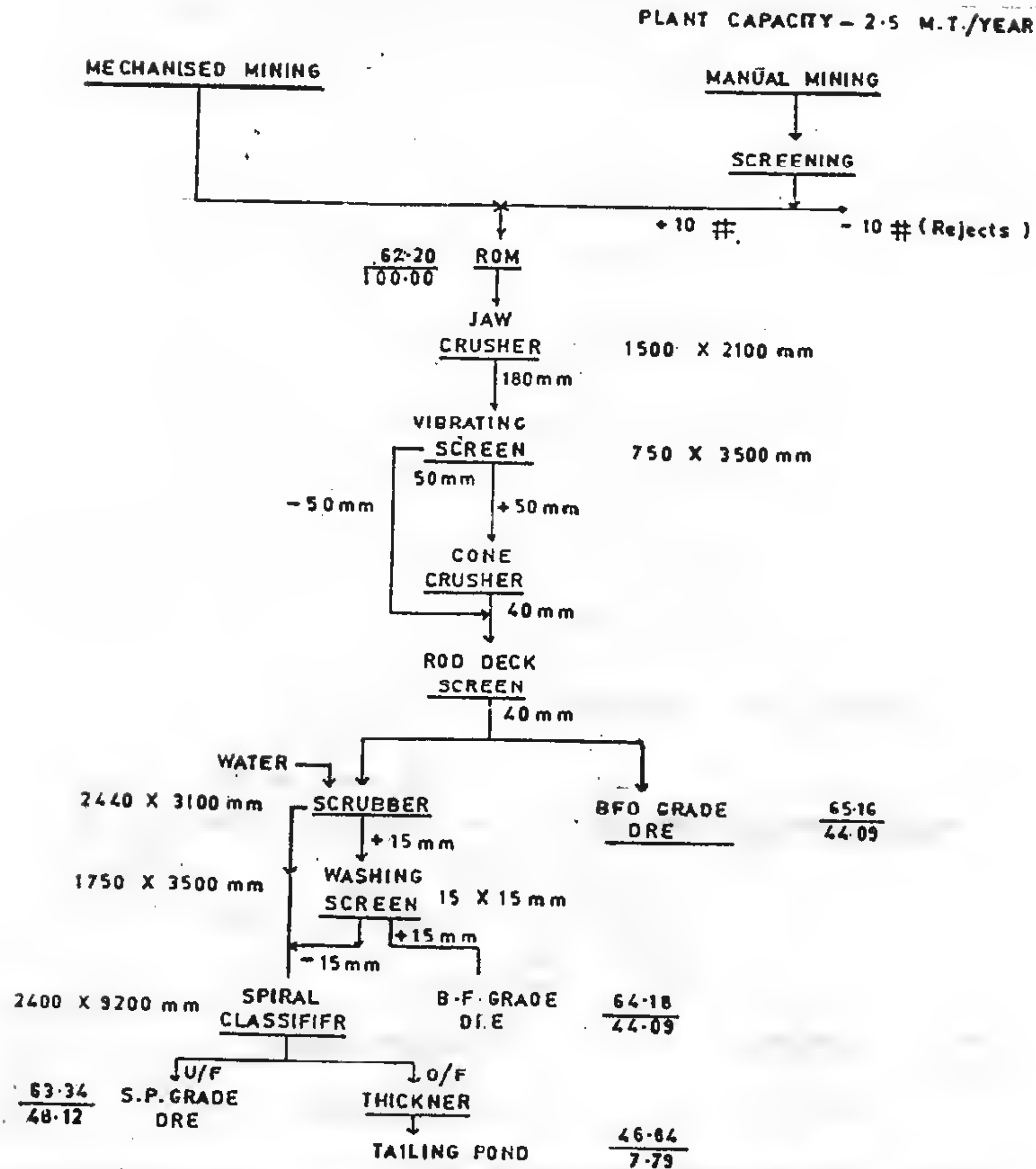
1 LOCATION – Iron ore washing plant at Bolani.

2 CHEMICAL ANALYSIS — (Average)

PRODUCT	% Fe	% Dist.	% Al_2O_3	% SiO_2	% MgO
Feed	61·5	100·00	5–7·5	3–6	2–5
Lumps	60–61	10·00	4–6	2–6	2–5
Washed Fines	62–63	60–65	4·3	3–3·5	2–3·5
Slimes	48–49	25–30	12·2	10–12	3–7

3 . PROCESS — Crushing, Screening, Scrubbing, Classification.

Fig. 10.6 : Flowsheet of Iron ore (Hematite) Beneficiation Plant of M/s Dalli Mines of Bhilai Steel Plant, M.P.



1. LOCATION – The plant is located at Dalli mines Bhilai M.P.

2 CHEMICAL ANALYSIS —

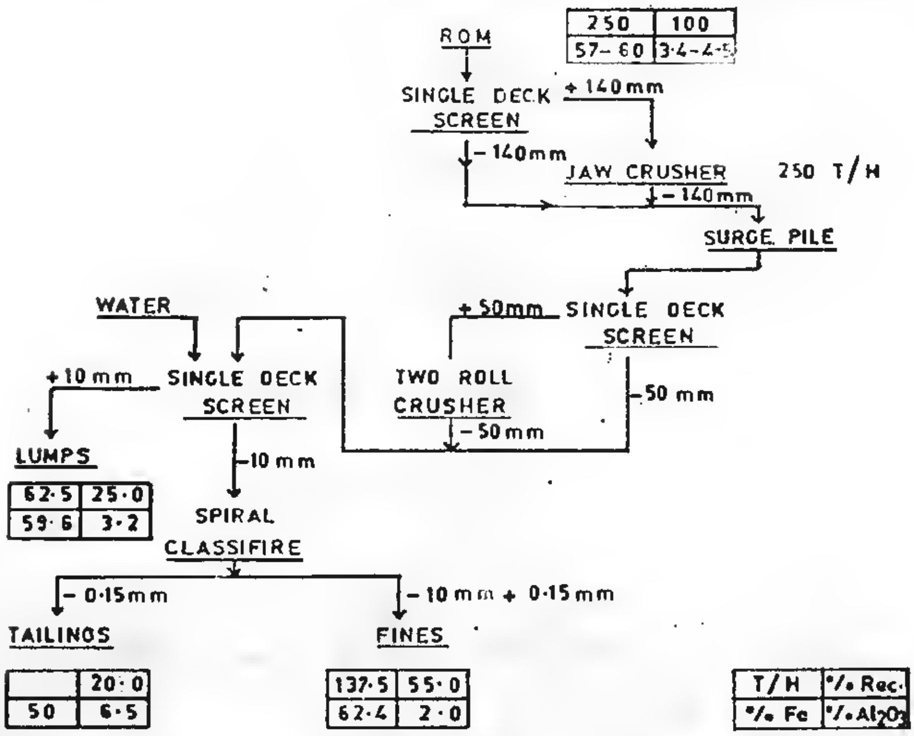
PRODUCT	Fe %	% Dist.	% SiO_2	% Al_2O_3	P
ROM Ore	62·00	100·00	3·60	4·60	0·057
B F O	65·16	44·09	1·88	2·54	0·055
B F	64·18		2·37	3·09	0·057
S P	63·34	48·12	2·82	3·69	0·059
Slimes	46·66	7·79	12·84	14·52	0·062

3 PROCESS – Crushing, Screening, Washing and Classification.

Fig. 10.7 : Flowsheet of Pale Iron Ore Beneficiation Plant of M/s Chowgule & Co. Ltd., Goa.

TWO PLANTS – A) DRESSING PLANT
B) WASHING PLANT

A – DRESSING PLANT CAPACITY – 250 T/hr



1 LOCATION – The plant is located near Pale village of Bicholim taluka of North Goa.

2 CHEMICAL ANALYSIS ––

PRODUCT	% Fe	% Dist	% Al_2O_3
Feed	57 – 60	100	3·4 – 4·5
Lumps	59·60	25	3·2
Fines	62·40	55	2·0
Tails	50·00	20	6·5

3 PROCESS – Crushing, Screening, Classification.